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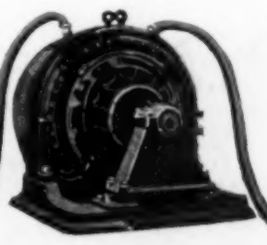

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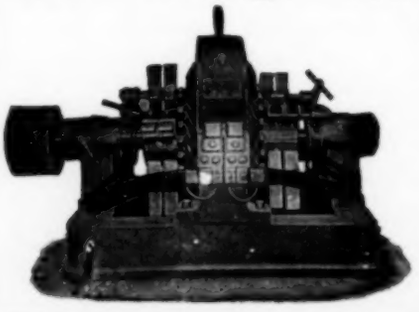

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The Montana Meeting of the American Institute of Mining Engineers

The 105th meeting of the American Institute of Mining engineers, held last month in Great Falls, Butte, and Anaconda, has turned out such a brilliant success as to leave no doubt of the beginning of a new era in the affairs of the Institute. The management of the meeting by the local committees, led by the officials of the Anaconda Company, was grand. The visits to the magnificent mines and smelters of the district were most instructive. The program of technical papers presented at the meeting was simply classical.

All the papers had been printed in advance. As the report of the meeting in this issue shows, the papers were of an exceedingly varied character, as they covered smelting and leaching, concentration and mining and geology. And yet they were a unit, as they all related to the Great Falls, Butte, Anaconda district. And they were all authoritative, being written by the responsible mining and metallurgical engineers of the district themselves.

Western hospitality is nothing new. But we doubt whether it ever manifested itself in a more glorious professional way than in giving to the world this set of papers on mining and metallurgy in Montana which will be the lasting monument of a great convention.

The Influence of South Dakota in Cyanidation

With the passing of time and the general adoption of successful processes and machines, the history of metallurgical development is likely to be lost to sight and current practice taken as a matter of course. And yet the historical feature of metallurgical progress is of surpassing interest, not only on account of its general bearing on the industry, but as showing how true it is that necessity is the mother of invention. In cyanidation, some of the most important steps have been evolved in obscure mills, where failure would have excited little more than local comment, but where success was of momentous importance to the operators. Subsequently the machine or method evolved has been recognized as fundamentally important, and its use has spread rapidly.

The closing article in the series on cyanide practice in South Dakota, appearing on another page, recalls the fact that the metallurgists of the Black Hills district have made many notable contributions to the art of extracting gold and silver from ores by the cyanide process. That district was the scene of some of the earliest attempts at cyanidation, by the American company which exploited the MacArthur-Forrest patents. The insuperable obstacles then encountered and the failures which ensued seem insignificant in the light of present practice; but they served to stimulate invention, and their record is easily forgotten in the consideration of the success which followed.

The importance of sand-slime separation was early recognized, for the first attempts at leaching unclassified pulp were decided failures. About 1899-1900, crushing in cyanide solution was introduced by John Henton, at Central City, in the first mill to operate continuously and successfully. Sand was separated from slime in spitzkasten or other form of settling box. The initial success of this step led to the erection of other mills on a similar plan, one of which for a long time held the record for low-cost operation in the Black Hills. Cones, spitzkasten, and settling boxes arranged for intermittent discharge of sand subsequently came into common use in cyanide mills for sand-slime separation.

In 1904 the separation of sand and slime was further improved by the invention of a mechanical classifier by J. V. N. Dorr who, in July of that year, operated the first of these machines in the mill of Lundberg, Dorr & Wilson, at Terry. A system of double-cone classification had been in use, but was inefficient on the kind of ore under treatment. It was impossible to obtain a leachable sand, free from slime, and failure of the mill appeared inevitable unless the separation could be improved. It was under such conditions, to avoid personal loss, that mechanical classification was developed and introduced into western cyanide practice.

It was in that day, also, that evolution in slime treatment began. Intermittent decantation had been standard practice, but about 1901 John Randall introduced continuous decantation with counter-current washing at the Lexington mill. Large cones were used, but were found poorly adapted to the purpose on account of irregular operation and the tendency of solid slime to build up on the sides. Despite the failure of the process at that time, it was recognized as embodying advantages over the old decantation process.

Although vacuum filtration of slime was not a contribution from South Dakota, the first Moore process plant to operate continuously and successfully was installed in the Lundberg, Dorr & Wilson mill in 1904. The slime was prepared for filtration by thickening in cones, but the inefficiency of this method again led to the development of a new machine. It was in 1906, at the Mogul mill, at Pluma, that Mr. Dorr invented and first operated a continuous, mechanical slime thickener. This machine has since found wide acceptance, not only in cyanidation where it has contributed to the success of continuous decantation, but also in concentration processes, particularly flotation.

Two other steps toward modern cyanide practice were first taken in the Black Hills, at the Homestake mills. For many years the metallurgical process employed here included only amalgamation and the leaching of sand with cyanide solution. Slime was a waste product. The loss represented in the latter was subsequently converted into profit through the inventions of C. W. Merrill, who devised the sluicing filter-press and the system of zinc-dust precipitation. The first large plant embodying these inventions was erected for the treatment of Homestake slime at Deadwood about 1906-7. Since that time the Merrill processes have enjoyed world wide use and have contributed to the success of the cyanide process.

The whole record speaks for itself. It is doubtful if any single mining district in the world has contributed to cyanidation more methods and machines which have been as uniformly successful as those which had their inception in the treatment of Black Hills ores. Whether they were evolved of necessity to save their authors from financial loss, or to increase efficiency and add to profits already assured, they reflect great credit on a small district in a state of lesser rank among the mining commonwealths of this country.

The Metal Markets in the Hot Months

In the past six months we have witnessed several pronounced phenomena in the metal business. First of all, an increasing market demand for each metal in face of an increasing price; second, an inhibition of real demand coupled with a tightening of credit in the world money-markets; third, a decrease of production concomitant with decreased demand, and, fourth, a holding of prices at reasonable levels by reason of the closing down or restriction of certain large producers. All this is typical of the primary and secondary metals and has its proximate cause in the great ease and distinct certainty with which intelligence is diffused among all classes in the second decade of the twentieth century. It is a hopeful sign of prosperity for the next few months.

Copper exemplifies this well, and so does iron and steel, spelter and lead. The first mentioned is as necessary to the age inaugurated by Faraday and Edison as water is to animal life. The second is as necessary to the age of the great engineering activity as air is to animal life. On these two metals depends our wonderful modern civilization.

General business, which is closely tied up in this age with the metal markets, shows a marked correspondence with the metal business. The buying movement in the latter part of the winter spread through all lines of the industrial fabric and all metals reached new high-level records of production. Latterly, the Balkan imbroglio, the trouble in Mexico, the uncertainty about the tariff and currency in the United States, the prodigality of government and private expenditure, the social unrest and ferment, all have been restricting factors, and have put the check rein on credit all over the world, but especially in Germany. Probably the uprising in the Balkans has had much more to do with the tightening of credit than is superficially apparent. This strain on credit has been real in its existence and potent in its influence. Its source was the great extravagance so manifest in the United States and in all other places in the world.

Among business men, the point of interest is now in the crops. While, if these are of great size, a temporary strain on credit will be produced at the close of 1913, yet the banks have generally so put their houses in order that no serious trouble can ensue. On the other hand, if crops are meagre a sharp period of general liquidation will cause hard times comparable to those of 1893. At present writing a large winter wheat crop has been harvested and the promise for spring wheat, corn, oats, and cotton is good. Quotations for these staples in Chicago are gradually descending. However, it must be remembered that two weeks of hot, dry, windy weather in the Mississippi Valley would "send these prices kiting." But the general diffusion in agriculture of what is known as scientific farming has so improved

the methods of the practical farmer as to slowly increase the yield and quality of foodstuffs that the chances are better than ever for large and bounteous crops.

It can be laid down as a fact, therefore, that the metal business must find its lead from the farming business and that until we know what the crops really are nothing but a continuation of present conditions can be looked for. In general connection with all this let us state that there are good judges of business who state that there must be a general liquidation and readjustment before any period of improvement or prosperity can start. Liquidation that should have come in 1908-1909 was tempered by certain powerful interests in a spirit of what we believe to be mistaken altruism. These same judges believe that we are starting on an era of descending prices following a fifteen-year period of ascending prices. Such a readjustment, though of benefit to all, will not be regarded as acceptable by the younger generation of energetic business men. All this is for the future. Right now a spirit of caution is the right one to have until the crops are gathered into the barns, and until Congress has produced something satisfactory or at least definite in the shape of a tariff bill and a currency bill.

The Output of Rolled Iron and Steel

Statistics of the output of rolled iron and steel furnish a more accurate index to our production of commercial material than do the more frequently cited statistics of steel ingot output. In the first place, the steel ingot is not an article of commerce, and indeed the production of steel ingots is not really the production of steel at all, but merely the production, by casting, of ingots. In the conversion of ingots into merchantable forms of steel much scrap is produced, which goes through, rather than into, the steel-making process again. In the second place, wrought iron is practically interchangeable with soft steel for many uses, and at any rate it is a closely cognate product, and in studying the tonnage output of rolled steel the tonnage of rolled iron may conveniently be added, furnishing an aggregate of the commercial material produced in rolled form.

Comparing the tonnage of steel ingots with the tonnage of rolled steel, one finds a fairly constant loss of approximately 24 per cent. This was the case in 1904 and 1905, back of which the present statistics of rolled steel do not go, and was also the case in 1912, the rolled iron and steel statistics of which year have just been presented. In a few of the intervening years the loss dropped to 22 or 21 per cent., but these were years of light production when ordinary proportions were disturbed.

Practically none of the material thus subtracted is actually lost. A relatively small portion of the steel is converted into scale, useful in the blast furnace and in some other directions, while by far the major part becomes new steel scrap, used in the open-hearth furnace. Before the general adoption of the basic open-hearth furnace such scrap was customarily charged in the Bessemer converter, being used in varying proportions to cool the heat and thus make up for variations in silicon content of the blast furnace metal. As steel works have found the better outlet for this material through the open-hearth furnace, the Bessemer department has been deprived of it, and the blast furnace is required to work more closely to silicon than formerly though the mixer has helped in this direction.

While the proportion of Bessemer steel to total steel has been decreasing constantly, it was nevertheless one-third in 1912, and with a loss of 24 per cent. from the ingot to the finished rolled steel, and further production of scrap by processes subsequent to the final hot rolling which marks the point at which a given material in course of manufacture is returnable for the statistics of "rolled steel," makes a total of scrap produced which it can readily be seen is quite large in proportion to the output of open-hearth steel. Thus a large percentage of "scrap" can be used in the open-hearth process without a great deal of old material being consumed. There is a somewhat popular misapprehension that the open-hearth steel industry consumes a large tonnage of old material, which is not an accurate conception. Detached open-hearth plants do use a large percentage of old material, but the basic open-hearth plant which draws direct metal from its blast furnaces through the mixer, and also draws scrap produced in rolling Bessemer steel from an attached plant, can get along without any outside scrap, and only employs it when it can be secured at inviting prices.

The following table shows, for the past seven years, our output of steel ingots (not including castings), rolled iron and rolled steel, in gross tons:

	Steel ingots.	Rolled steel.	Rolled iron	Total rolled.
1906.....	22,624,431	17,401,911	2,186,557	19,588,468
1907.....	22,559,477	17,664,736	2,200,086	19,864,822
1908.....	13,677,027	10,589,744	1,238,449	11,828,193
1909.....	23,298,779	17,935,259	1,709,431	19,644,690
1910.....	25,154,087	19,881,123	1,740,156	21,621,279
1911.....	23,029,479	17,578,556	1,460,615	19,039,171
1912.....	30,284,682	23,019,259	1,637,582	24,656,841

The reported output of rolled steel represents the entire commercial material produced from the steel ingots, each weighed after its last hot rolling (including necessary shearing), thus including skelp rather than pipe, rods rather than drawn wire, sheets and black plates rather than sheet bars, and so on. For 1912 exported slabs, billets, etc., are included, but for earlier years this desirable inclusion was overlooked.

The decadence of rolled iron is striking, the year 1907 having furnished the maximum tonnage in recent times. Prior to the general adoption of mild steel there was a slightly larger tonnage of rolled iron, the entire industry being so small, however, that the maximum tonnage seems to have been about 2,500,000 tons, in 1890.

There is a very different alignment in the relative tonnage of different rolled products than obtained only a few years ago. The output of rails, for instance, reached its maximum tonnage in 1906, since when there has been an absolute decline in the tonnage and, of course, a still greater decline in the relative tonnage. Rails in 1912 comprised less than 14 per cent. of the total rolled product, merchant bars comprising 15 per cent., while the tonnages of rods, plates, sheets (including black plates for tinning) and structural shapes all lay between 10 and 14 per cent., and skelp fell just short of 10 per cent. Thus we have seven products, rods, rails, shapes, plates, skelp, sheets and merchant bars, whose tonnage is not greatly dissimilar These seven products in 1912 comprised 83.3 per cent. of the total rolled iron and steel output of the country.

Readers' Views and Comments

Formulæ for Calculation of Alkali Hydroxides and Carbonates

To the Editor of Metallurgical & Chemical Engineering:

Sir:—The analytical chemist is often called upon to determine the amounts of alkaline hydroxides and carbonates present in a sample of NaOH or KOH. The method which calls for the least amount of time is the following: Double titration in the cold with phenolphtalein and methyl orange according to the method of R. B. Warder (Treadwell & Hall, Quant, Anal.), together with the determination of the sodium and potassium salts according to the indirect analysis method.

It can be said of the indirect analysis method that it is not extremely accurate except in case the sodium and potassium salts are present in about equal amounts (or say 20% of one to 80% of the other), but it must be remembered that we are not here concerned with the determination of small amounts of one in presence of the other.

Under the conditions mentioned these methods are accurate and are shorter by far than any others as far as the mere operations are concerned, but the calculations are long and tiresome in the extreme. It was to shorten the latter that the writer worked out the following formulae. These have been checked against hypothetical mixtures and will be found to be accurate within 0.01%.

ALKALINE HYDROXIDES AND CARBONATES ON BASIS OF ONE GRAM CHARGE FOR ANALYSIS

$$\% \ NaOH = \frac{0{,}0561\ (T - t)}{0{,}02805 + 0{,}02109\left(\frac{N}{M}\right)}$$

$$\% \ KOH = \frac{2{,}1093\ (T - t)}{\left(\frac{M}{N}\right) + 0{,}7840}$$

$$\% \ Na_2CO_3 = \frac{0{,}1831\ (t)}{0{,}03455 + 0{,}02709\left(\frac{N}{M}\right)}$$

$$\% \ K_2CO_3 = \frac{5{,}4176\ (t)}{\left(\frac{M}{N}\right) + 0{,}7840}$$

Where,

T = No. cc. half normal acid to phenolphtalein as indicator.

t = No. cc. half normal acid to methyl orange as indicator.

M = Weight NaCl found in indirect analysis.

N = Weight KCL found in indirect analysis.

H. A. Mereness.

Knoxville, Tenn.

The Ostwald Process for Making Nitric Acid from Ammonia and the Production of Ammonia from Calcium Cyanamid

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The article in the August issue on this subject (page 438) is of great interest to me. It is true that the process of making nitric acid from ammonia produced from calcium cyanamid is new for England as stated there. But exactly this same process has been and is in successful commercial operation in a plant in Belgium though nothing has yet been published about it.

The following notes on the process should, therefore, be of interest. Naturally certain important details are being kept secret and must be withheld from publication.

If we mix calcium cyanamid with water we get ammonia.

$$CaCN_2 + 3H_2O = CaCO_3 + 2NH_3.$$

To mix cyanamid with plain water is impractical for industrial purposes for the reason that not all of the nitrogen can thereby be extracted in form of ammonia. In order to obtain all the nitrogen as ammonia, we must use alkali and carry on the process along the following lines which have been developed in many years of hard work and experimental research.

If we mix the correct quantities of $CaCN_2$, H_2O, Na_2CO_3 and NaCl in one dish and mix well, acetylene will be evolved. This is due to the fact that cyanamid contains 0.1 to 0.3 per cent of carbide of calcium. After the carbide has thus been removed the mixture is transferred to an autoclave and the steam turned on and the pressure raised to 6 or 8 atmospheres. All the nitrogen will then distill in the form of ammonia gas.

From my own experience I know that this distillation goes so well and so quickly that in ordinary daily operation one autoclave will distill in about one-half hour the ammonia from 1000 kg. cyanamid which contains 20 per cent of nitrogen. The extraction is nearly complete, as will be seen from a few analyses of the cyanamid before and after the distillation.

% Nitrogen in Calcium Cyanamid

Before	After	Before	After
18.50	0.27	19.50	0.26
18.75	0.27	19.75	0.34
19.00	0.32	20.00	0.40
19.25	0.34	20.25	0.38

If the operation in the autoclave is carried out properly, the ammonia and steam come out entirely clean without carrying any cyanamid dust. Nevertheless, the practice is to pass the ammonia and steam through a cleaning apparatus for the purpose of catching any dust which might be carried along.

After that operation ammonia and steam are passed together into a column which cools and condenses the steam and lets pure ammonia gas enter into the gasometer.

Ammonia obtained by this process is very pure, the purity being 99.8 per cent or more.

The entire process is easily carried out and is a large source of ammonia which may be transformed into many chemical substances. We can transform ammonia nitrogen into nitric nitrogen from which we can get very easily ammonium sulphate or liquid ammonia, but the most interesting transformation process is the transformation into nitric acid and ammonium nitrate.

If we let the ammonia pass through tubes containing a catalytic agent (platinum) we can transform ammonia H_3N into NO_2 and with further oxidation in oxidation towers into nitric acid of 36 deg. B. This acid we can transform with the same ammonia into nitrate of ammonium and we can continue concentrating further to highest concentrations.

The process is exceedingly interesting and very simple and in the writer's opinion it has a great future. A. B.

The Western Metallurgical Field

Refining Sulphur in Wyoming

About three miles west of Cody, Wyo., is a large deposit of native sulphur, varying in thickness from 60 to 75 ft., and lying from 1 to 7 ft. below the surface. Recently the Northwest Sulphur Co. has secured control of 1,400 acres of this sulphur-bearing ground, and has tested the deposit by means of 150 drill holes. The results of drilling indicate the presence of about 500,000 tons of crude sulphur.

A part of the deposit has been uncovered, and sulphur is being mined from open pits. A small refinery has been erected on the property, and sulphur is being refined at the rate of about 12 to 15 tons per day. The system of refining

is extremely simple, and the equipment required is not expensive either in first cost or in operation. The plant consists essentially of steam boilers, retorts, retort-cars and bins. Figs. 1 and 2 give views of parts of the plant.

The crude sulphur is dumped into the retort-cars which are run into the retort and subjected to the action of steam at 60-lb. pressure. During this process molten sulphur runs from

FIG. 1.—RETORT-CAR, RETORT AND BOILER

the retort and into storage bins, and when a charge has been exhausted the car is withdrawn and dumped. The refined sulphur is reported to be 99.8 per cent pure.

During the month of July, 1913, the refinery produced an average of 12 tons refined sulphur per day, at a cost of $8.40 per ton.

As the product sells for about $20 per ton at Cody, Wyoming, the operation is decidedly profitable. In the month of August the capacity of the refinery was doubled, bringing it up to about 25 tons per day. Fuel cost is low, the company has about 200 acres of coal land from which a regular supply is drawn.

Jigging Black Sand and Gravel

Experiments now being conducted in Denver point to the possible use of the Richards pulsator jig in the retreatment of black sand and dredge gravel. The tests on the former were made with material collected in sluice boxes, and showed remarkable results. A small laboratory jig was used, bedded with gravel from ⅛ to ¼ in. diameter, and arranged to yield spigot, side-gate and tailing products. The black sand contained considerable clay, which apparently coated the gold particles and prevented amalgamation. Careful panning of the different products after each test showed that practically all the gold was obtained in the spigot discharge. In one or two instances a color was obtained by panning the side-gate discharge, but none was observed in the tailings. A further interesting point was that the gold thus obtained was readily amalgable, whereas before jigging it was not. Most remarkable of all, perhaps, was the complete recovery of very fine gold dust. It was to be expected that small particles of appreciable side would be recovered without difficulty, but it was surprising to note that the very finest dust also was collected in the spigot discharge, and that the other products were practically free from gold.

The application of the jig to dredge gravel is probably of greater importance. Tests on material passing a ½-in. screen suggest the possibility of jigging this size product, and obtaining a spigot discharge which can then be retreated on a second jig, with practically complete recovery of the gold. The arrangement of the jigs on the dredge boat should not be a difficult matter, as the machines are small, and the necessary power and water would be at hand.

Flotation Litigation at Butte

Mention has previously been made in these columns of the fact that the Butte & Superior Copper Co., at Butte, was using a flotation process under the patent of Mr. James M. Hyde. Recently the Mineral Separation Co. brought suit against the Butte & Superior Company, alleging infringement of the flotation process owned by the former. The case was heard and decided in favor of the Minerals Separation Co., and a decree was ordered as demanded by the plaintiff for treble damages and an accounting before a master in chancery. It is understood that the Butte & Superior Co. will appeal, and in the meantime operations will continue as in the past.

The history of litigation over flotation processes, as outlined in Mr. T. J. Hoover's book on "Concentrating Ores by Flotation," suggests the futility of trying to settle technical matters in courts of law. The record is one of decisions and reversals and compromises. As the author himself suggests: "All this mass of litigation is inconclusive. No one of these suits would involve a decision on all the patents. In fact, if all the law suits already started were brought to a final decision it is doubtful whether the public would know whether they could use the processes of successful litigants without fear of further litigation. If the whole situation could be placed before a competent court to decide on merit it would be far preferable to the engineer, but it might be less acceptable to the legal profession, for obvious reasons."

New Smelting Plant for the Granby Company

The Granby Smelting Company, operating in British Columbia, is constructing a new 2000-ton smelting plant at the Hidden Creek mines, Granby, B. C., at a cost of about $3,000,000. According to a statement by Mr. Jay P. Graves, vice-president and general manager for the company, construction is progressing rapidly, and on July 1 there were 1125 men on the pay roll. Structural steel for the smelter buildings is on the ground, a brick plant is practically completed, and railway connections have been established. With no unforeseen delays the plant should be ready for operation about January, 1914.

One of the features of the plant is the construction of the underground ore-storage and crushing plant, connected with the smelter by tunnel. This arrangement will effectually over-

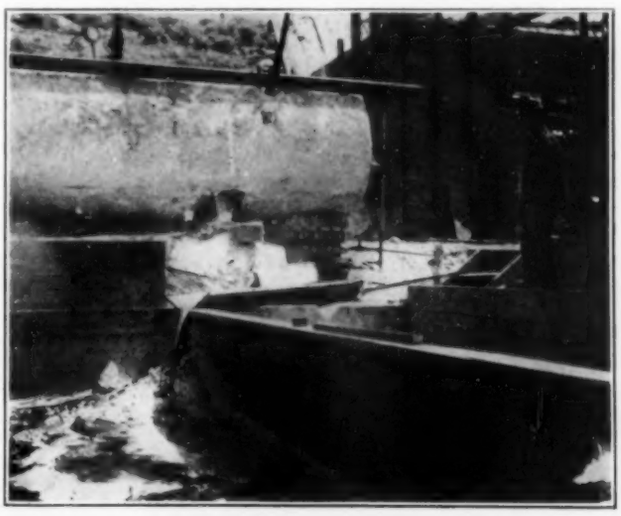


FIG. 2.—SULPHUR RUNNING INTO BIN FROM RETORT

come delays and difficulties of operation incident to the severe winter weather.

The company is establishing a model town and will build and maintain a recreation hall and gymnasium for its employees. A 40-room modern hotel is completed and a hospital will be built, comprising six private wards and one large ward, with operating room and every facility for caring for the ill and injured.

Labor Strike in Michigan

At the call of the Western Federation of Miners, the first general labor strike in the Lake Superior copper country occurred on July 23. This has been anticipated for a year past, during which time it has been understood that Federation workers were steadily organizing the miners of the district. Without other formality than the presentation of a protest against the one-man drill and a request for shorter hours and more pay, the strike was precipitated, and in a few days all mines and mills were closed. As a result of disturbances the state militia was called out and placed in charge of the upper peninsula of Michigan.

The specific demands of the Federation included recognition of the union, a minimum wage of $3 per day of 8 hours, and the abolishment of the one-man drill. To this request the operators made no reply, and since the strike was called they have steadfastly refused to recognize the Federation or treat with its officers, even at the request of Governor Ferris. They have, however, indicated their willingness to meet their men as employees; and in defence of their attitude have cited their consideration for the men during past years and their success in adjusting differences between employees and the companies. It is believed that the firm decision of the operators to ignore the Federation is taken mainly on account of the past unsavory record of that organization in other mining camps where similar strikes were inaugurated.

At last reports the strike was still on, with little hope of an early settlement of differences. Pumps are operating in practically all of the mines, but no underground work is attempted. Although there is considerable iron mining in northern Michigan, it is considered unlikely that any attempt will be made to spread the strike into the iron country, as the Federation is not strongly organized there.

Company Reports

Utah Copper Co.—The twenty-first quarterly report for the second quarter of 1913, states that the production of copper in concentrates was 31,785,448 lb., an average of 10,595,149 lb. per month. This was recovered from 1,910,214 tons of ore, of which the Magna plant treated 55 per cent and the Arthur plant 45 per cent. The mills averaged 21,000 tons per day. The average grade of the ore was 1.2807 per cent copper. The grade is below normal, due to the necessity of mining very low-grade and partially stripped areas. The average cost per pound of copper, after allowing for smelter deductions and without crediting miscellaneous income, was 8.933 cents. If miscellaneous earnings are applied the cost was 8.16 cents. While the cost per pound of copper was high, the cost per ton of ore treated was reduced to a figure lower than had been attained in any previous quarter. The cost of mining and milling, including charges of every nature, was 96.3 cents per ton. The total net profit for the quarter was $2,218,753, computed on the basis of 15 cents per pound for copper. Construction expenditures of consequence will have ceased by the end of the current quarter.

Nevada Consolidated Copper Co.—The sixteenth quarterly report, covering operations during the second quarter of this year, shows that the mill treated 762,880 tons of ore of an average grade of 1.76 per cent copper. The cost per pound of copper produced was 8.95 cents, as compared with 10.61 cents for the preceding quarter, and 8.33 cents for the last fiscal year. The net credit to undivided profits was $98,422, after paying the fifteenth quarterly dividend and making charges for depreciation and ore extinguishment. Earnings were computed on the basis of 14.392 cents per pound for copper. The contract existing between the Steptoe Valley S. & M. Co. and the Giroux Consolidated Mines Co. for the treatment of the ores of the latter company, has been cancelled by the Giroux organization and will become effective in June, 1914. The contract was for five years with the privilege of cancellation by either party on one year notice.

Ohio Copper Co.—A report made public under date of July 22, states that the construction of the third section of the mill has been completed and that the plant is now treating between 71,000 and 75,000 tons monthly, with a total cost of mining, milling and transportation of 85 cents per ton. The copper content of the ore has averaged 1.015 per cent. The earnings have not been satisfactory and the company is contemplating the use of flotation instead of the standard process now installed. With this in view, Mr. M. W. Atwater has been retained to make investigations.

The Iron and Steel Market

Production of finished steel was heavy in August, at between 85 and 90 per cent of full rated capacity of the mills, substantially the same rate as in July. As outputs are normally decreased by hot weather, the showing for the two months is very good from a commercial standpoint. Except for the normal stocking of wire products against the fall movement to distributors, all the steel produced is being shipped and as there is no suggestion that buyers are stocking—the tendency is rather to reduce stocks—the indication is that actual ultimate consumption has proceeded at a heavy rate.

Strictly new buying was somewhat heavier in August than in July but as it remained far below the depletion of mill order books proceeded farther and finally reached the stage, which it was certain several months ago would be reached at one time or another, of mills looking for additional business of importance at cut prices. At the close of August the steel market in general is in a transitionary stage, prices tending to decline to seek the level at which buyers will take hold more freely and actually fill up the mills. How far the readjustment will proceed cannot possibly be predicted, as it is a question of restoring a balance between elements which are not well understood. A great deal, too, will depend upon how the mills determine to shape their policy, whether to seek simply enough business to round out the year, delaying the greatest test until later, or conclude to book enough business to furnish a good foundation for running through the winter and into next spring. This is largely a matter of judgment and decision among a relatively few large steel producers.

Pig Iron

The pig iron buying movement came to a close early in August. In southern iron it was heaviest late in June and early in July; in northern iron it centered in July. The long decline, lasting from late in December until July, was clearly arrested, but only by the great help afforded through a material decrease in production, which decrease was spread over a period of several months, and amounted to a total of perhaps 20 per cent among the merchant furnaces. Had the buying movement lasted a little longer a definitely advancing market would undoubtedly have been inaugurated, by buyers being encouraged to purchase for deliveries farther ahead, but as it was most purchases were only through the third quarter, though a few were made through the half year, particularly in southern iron. The average level of quoted prices has been advanced, through the movement, but not over 25 cents, but taking cognizance of the fact that a number of important sales were made at lower figures than openly quoted the advance might be considered close to 50 cents. At the close of August the market is stagnant, but with the operating furnaces comfortably filled for two or three months, buyers being equally well covered. We quote: Southern No. 2 foundry at Birmingham, $10.75 for small prompt lots, with $11 quoted usually for such delivery, and firmly maintained for fourth quarter; No. 2 X foundry, Philadelphia, $16.65@16.75; No. 2 X foundry, f.o.b. Buffalo furnace, $14; No. 2 foundry delivered Cleveland, $14.50@14.75; No. 2 foundry at Chicago furnaces, $15; at valley furnace (90 cents higher, delivered Pittsburgh) No. 2 foundry, $14; malleable, $14.25; basis, $14@14.25; Bessemer, $15.75@16; forge, $13.50.

Steel

The fore part of August was very dull in unfinished steel, with former quotations maintained on billets at $26.50 and sheet

bars at $27.50, while rods dropped from $29 to $28. After the middle of the month sheet bars began to be offered at cut prices by the smaller mills while larger mills offered to make sliding scale contracts, based on the average price of basic pig iron to be disclosed monthly, which would net the sellers less than $25. At this writing the large mills seem to be reaching the conclusion that they will have to accept much lower prices and it seems probable that in the early part of September billets can be obtained at $23 or $24 and sheet bars at $24 or $25, maker's mill, Pittsburgh or Youngstown, while rods are likely to remain at about $28, Pittsburgh.

Finished Steel

As already indicated there is a distinct declining tendency in finished steel prices, touching a large number of commodities, if not the majority. The tendency is not well shaped at present, and may involve only slight reductions, or may eventually involve very considerable reductions. Meanwhile we quote former prices on plates, bars and shapes, though some of these products have already been rather openly shaded. Sheets have definitely declined further, in that prices formerly quite exceptional are now made much more generally. Wire products have declined $1 a ton. Butt weld sizes (3-in. and under) of steel pipe have been reduced one point or about $2 a ton. We quote prices as follows, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Common iron bars, 1.60 cents, Pittsburgh; 1.25 cents at eastern Pennsylvania mill; 1.35 cents, delivered Philadelphia; 1.40 cents, Chicago.

Wire nails, $1.65, base; plain wire, 1.45 cents.

Sheets, blue annealed, 10 gage, 1.65 cents; black, 28 gage, 2.15@2.20 cents; galvanized, 28 gage, 3.20@3.25 cents; painted corrugated, 28 gage, 2.35@2.40 cents; galvanized corrugated, 28 gage, 3.25@3.30 cents.

Merchant steel pipe, 80 per cent off list for ¾ to 3-in.

Steel boiler tubes, 3½ to 4½-in., 69 per cent off list.

Standard railroad spikes, 1.70 cents, Pittsburgh; 1.75 cents, Chicago.

Button head structural rivets, 1.90@2 cents; cone head boiler rivets, 2@2.10 cents.

The Non-Ferrous Metal Market

Since our last report metal prices have tended generally higher, and business has been transacted on a fairly large scale. The labor strike in the Lake copper country and threatened strike in the lead district of southeast Missouri have reacted in favor of higher prices. Spelter has been scarcer on account of the closing of some western properties at the time of the recent slump in the market, and prices for this metal also have been raised.

Copper.—Since the Lake strike, prices for Lake copper have been nominal and sales have been on a very small scale. The electrolytic market has been firmer and this brand of copper is becoming scarce for immediate delivery, although no premiums for spot metal have yet been asked. Lake is quoted nominally at 15⅞@16 cents and electrolytic at 15.50@15.60 cents.

Tin.—Domestic demand has been increasing and large orders have been placed abroad. As a result spot metal is commanding a premium of about ½ cent and the quotation for August tin was 41½ cents.

Lead.—Prices for this metal have advanced since our last report, and early in the month the market was active and strong. Later it was quiet and without special feature, with quotations at 4.37½@4.40 cents, St. Louis, and 4.45@4.50 cents, New York.

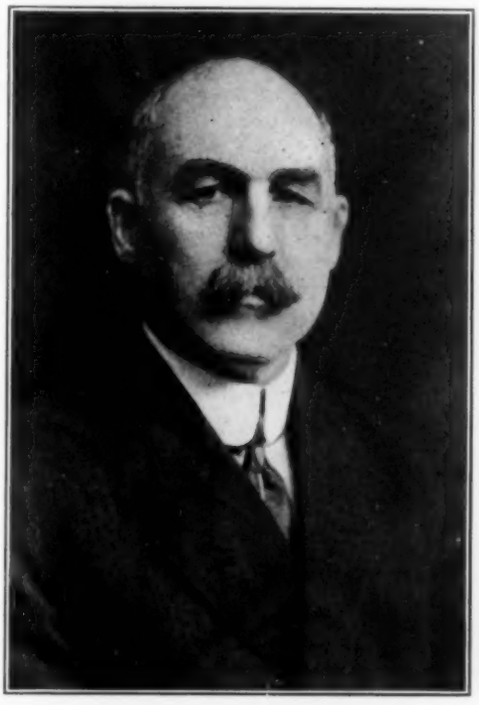
Spelter.—A good demand has been in evidence and higher prices have been paid. Production is still below normal, owing to reduced output from western mines. The last quotations are 5.45@5.50 cents, St. Louis, and 5.60@5.65 cents, New York.

Other Metals.—The aluminium market is rather dull, and sales are only for immediate demands. Current quotations tend lower, being 22¾@23¼ cents, New York. The antimony market also is dull, with sales of small volume. Various brands are quoted at from 7.25@8 cents. Business in quicksilver is fair and prices are unchanged at $40 per flask of 75 lb., New York, and $39.50, San Francisco.

Eugene A. Byrnes

Dr. Eugene A. Byrnes, of Washington, D. C., a distinguished patent attorney and expert, died on August 1st at Haven, Maine, whither he had gone for his vacation. He died of acute peritonitis following an attack of appendicitis.

Dr. Byrnes was born in New York State in 1862. He graduated from the University of Michigan, and in 1884 came to this city, where he was for a time a teacher in the Central High School. Subsequently he entered the Examining Corps



THE LATE EUGENE A. BYRNES

of the United States Patent Office, and by his industry and ability rose to the position of Chief of a Division, being meanwhile graduated from the Law School of Columbia, now George Washington University, and was admitted to the Bar. He also took a post-graduate course leading to the degree of Ph.D. in electrochemistry and physics. In 1901 he resigned his official position, and entered upon the practice of his profession, taking as a partner Mr. C. P. Townsend, and later, also Mr. J. H. Brickenstein, the three constituting the well-known firm of Byrnes, Townsend & Brickenstein.

From the resolutions passed by the Patent Law Association of Washington on the occasion of Dr. Byrnes' death we quote the following:

"Modest and unassuming, but of a studious nature and with a fondness for scientific investigation, Dr. Byrnes was promptly accorded high standing among his professional brethren, by reason of his ability and attainments. He occupied the unique position in the patent profession of being at once a lawyer of ability and an expert chemist and physicist, whose opinion and advice were sought in matters of this character by leaders of the profession, and unhesitatingly accepted.

"Death claimed him in the prime of vigorous manhood, and with scant warning. Those whose privilege it was to come into contact with Dr. Byrnes, whether in a business or a social way,

bear grateful testimony to his unfailing courtesy and kindliness, and to the fairness and candor of his nature. In his death this community loses a worthy citizen, the profession and this Association a fine example, and his business associates a loving comrade, and a strong and able co-worker. Words cannot express the loss to his wife and relatives."

This journal loses in Dr. Byrnes a good friend and ever faithful contributor. Right from the start of this journal in 1902 Dr. Byrnes has prepared the "Digest of United States Patents" (prior to 1902), arranged according to subject matters in chronological order. A little more than a week before his death we received from him his last manuscript. His Digest, a work of love and a by-product of his professional activity, survives him and to his many friends in the electrochemical fraternity who have used it and will use it, it will serve as a pleasant remembrance of the fine personality of Dr. Byrnes.

William James Evans

In the death of Mr. William James Evans, on July 23, New York chemists and the Chemists' Club have suffered a serious loss. Although Mr. Evans had been ill for some time he had partially recovered and his many friends in the Chemists' Club were delighted to see him able to visit the Club again. His death followed an operation which suddenly became necessary owing to a return of his former trouble.

Mr. Evans was born in Port Wilson, Ontario, Canada, August 31, 1852. In July, 1880, he entered the employ of Messrs. McKesson and Robbins, and was actively engaged as

THE LATE WM. J. EVANS

a salesman for some years; later he took up detail work among the physicians and druggists of New York City. Mr. Evans was a member of the American Chemical Society, of the American Pharmaceutical Association, of the Society of Chemical Industry, and of the Lotus and Drug and Chemical Clubs. He was the past president (1910-1911) of the Canadian Society of New York. He was an active member of the Schools and Universities Club of New York, and was the ever jovial president of the Fendsophs.

Mr. Evans joined the Chemists' Club on March 13, 1908, and was elected treasurer the following December. He served as treasurer during the years 1909, 1910, 1911, and 1912, and was particularly active in putting the club on a sound financial basis during its most critical period before and after the removal to the new building in Forty-first Street. In his own quiet and jovial way Mr. Evans has done an immense amount of good for the Club that shall never be forgotten.

From resolutions adopted by the Board of Trustees of the Chemists' Club we quote the following:

"The kindly and friendly interest which Mr. Evans took in the Club and all its functions, his sincere and cordial meeting with all its members, and his efficient and unflagging cooperation in forwarding the interests of the Club made him not only a member of unusual usefulness but also a warm personal friend of each of us."

Everyone who knew him—and this means every New York Chemist—knows how true this is.

Mr. Evans leaves a widow, one son, and two daughters.

Personal

Dr. **Richard Beck,** president of the Freiberg Bergakademic, of Freiberg, Saxony, will be the guest of honor at a dinner of the Old Freibergers in America at the Engineers Club in New York City. Further particulars may be obtained from O. L. Bryden, 701 Jefferson Avenue, Scranton, Pa.

Mr. **J. E. Johnson, Jr.,** has opened an office at 52 William Street, New York, as consulting engineer and metallurgist. He will devote himself to investigations and reports on iron and steel lines as well as to design and operation problems and to expert work in patent causes. Mr. Johnson has been connected for a good many years with iron and steel works, involving large power and metallurgical operations and has carried on important investigations. The more recent of these aimed at determining the causes of differences in quality of pig irons of the same analysis. Mr. Johnson will be associated with Messrs. Sanderson and Porter, engineers and contractors.

Mr. **Milton M. Kohn** has returned to New York City, from his European trip.

Mr. **Philip J. Kroll** has been appointed sales agent of the International Oxygen Company, manufacturers of oxygen and hydrogen generators of the I. O. C. system, for the Pittsburgh district. Mr. Kroll's headquarters will be at Room 1023, Park Building, Pittsburgh

Professor **Albert Sauveur** of Harvard University has been awarded by the Franklin Institute of Philadelphia the Elliott Cresson Gold Medal, the highest award in the gift of the Institute "in recognition of his numerous and important contributions to the science of metallography and the influence he has exerted in bringing this science into practicable and exceedingly useful application in the iron and steel industry."

Mr. **Edson O. Sessions** has opened an office as consulting engineer at 5022 Sheridan Road, Chicago.

Mr. **Frederick W. Sperr, Jr.,** until recently chief chemist of the by-product coke plant of the Tennessee Coal, Iron, and Railroad Company at Fairfield, Ala., has resigned to take charge of the blast furnace and by-product laboratories of the Inland Steel Company, at Indiana Harbor, Ind.

The First Mining Show and Industrial Exposition, to be held under the auspices of the American Mining Congress, will be held at the Horticultural Hall, Philadelphia, Pa., from October 17 to 25. Full information relating to the Exposition may be obtained from Richard L. Humphrey, Director, 1031 Land Title Building, Philadelphia, Pa.

The Multiple Unit Electric Company announces their removal from 136 Liberty Street, to 133-7 East Sixteenth Street, New York City, which is necessitated by an increased volume of business. In the new quarters will be installed improved labor and time saving machinery, which will greatly facilitate the production of their electric furnaces and heating appliances. While the company's products have received the flattering approval of the technical public, owing to their simplicity, durability, and efficiency, the company realizes that it was possible to improve upon their product, and one of the first moves in this direction was to air-cool the electrical connections and place them in position far removed from the heat zone. This means that the furnace proper need never be disturbed, and a simple removal of a heat unit produces a practically new furnace. There has been a change in the personnel of the company, and Mr. O. A. En Holm has been appointed general manager with complete supervision over office and manufacturing departments; Mr. M. M. Kohn being no longer connected with the company.

Evolution in Methods of Handling Slime.

VI.—Rhodesia

By H. N. Spicer

Rhodesia is probably one of the oldest gold-mining countries in the world, history proclaiming that King Solomon drew some of his gold supplies from that region. Nevertheless, today the mines of Rhodesia are profitable and far from exhausted;

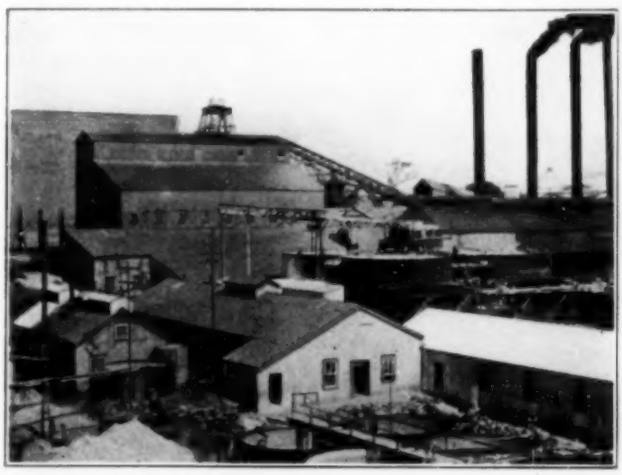


FIG. 1.—GENERAL VIEW GLOBE AND PHOENIX 40-STAMP MILL.

and from a metallurgical point of view the country is considerably more up-to-date than any other in South Africa.

Until within the last few years, Rhodesia has been regarded as a country for the prospector and the small company content to operate a small mill on a comparatively high-grade ore. More recently, however, owing to the development of some of the mines, and to the fact that several of the large Transvaal mining houses have interested themselves in the country and have acquired what appear to be the best of the mines, work has been undertaken on a large scale.

Mills of 1000 to 2000 tons daily capacity are now in course of erection.

Influence of Australian Practice

The development of cyanidation in Rhodesia has been greatly influenced by Australian practice, due to the fact that some years ago metallurgists and engineers from the latter country found their way to Rhodesia. Naturally they brought with them the ideas which they had developed and used successfully in Kalgoorlie and other Western Australian camps.

As a whole, Rhodesia is a dry country, though not nearly so dry as Western Australia. As a consequence the water problem is serious; and one of the main factors in successful milling is the use of a minimum of water and the conservation and re-use of that element. The Australian metallurgist was well qualified through long experience to tackle the problem in Rhodesia, making extensive use of plate-and-frame presses and exhaust-steam condensing plants. The present use of these devices in Rhodesia points to the prior education of the men who installed them.

Globe and Phoenix

The premier gold mine of Rhodesia is the Globe and Phoenix, from which handsome profits have been derived for a number of years. For a long time this property was considered the one exception to the belief that the ore-bodies in Rhodesia, though rich, were small and did not extend to any great depth. This mine is now being worked at a depth of nearly 2000 ft., and the published report of the company states that the ore-body continues to hold its own in width and value.

The Globe and Phoenix company has two mills: one for the treatment of ore directly from the mine, and the other for the retreatment of the accumulated tailings from the first mill.

In Fig. 1 is given a general view of the first mill. In the left background is the building for water recovery by steam condensation. Directly in front of that is the 40-stamp mill, with the power-house at the right.

Forty 1250-lb. stamps crush the ore in water through 20-mesh screens, averaging about 6 tons per stamp in 24 hours. The average gold content of the ore milled is 1 oz. per ton. The crushed ore is first amalgamated and then ground in twelve 5-ft. Forwood-Down Australian grinding pans. The pan discharge is roughly classified into sand and slime in small spitzkasten; the sand is concentrated on Record vanners, making a product representing about 4 per cent of the total ore treated. Both sand and slime are passed over blankets to recover coarse gold, after which they are elevated by means of a double-acting 12-in. vertical plunger pump and delivered to a series of large wooden spitzkasten for final classification.

Slime Is Weathered Before Cyaniding

The underflow from the spitzkasten flows to four 30 ft. by 8 ft. leaching vats where the sand is drained and treated with cyanide solution. The slimy overflow from these vats is combined with the overflow from the spitzkasten and sent to dams for settlement. Water is recovered and returned to the mill. The slime has a value of $4 per ton; but owing to the presence of a small amount of antimony it has not been possible to treat this material by cyanidation without first allowing time for oxidation by exposure to the atmosphere.

After the dumps have weathered for six months, during which time they are plowed and harrowed, they are treated by standard intermittent decantation at the rate of about 100 tons a day. The slime is mixed with cyanide solution in a vortex mixer, about 5 tons of solution being added to 1 ton of slime. Lime also is added at the mixer, at the rate of 4 lb. per ton. No other agitation is given than that received during the mixing operation, and the pulp is sent to two 35-ft. slime collectors. This slime does not settle readily, although 60 per cent of it is of a sandy nature. Two treatment vats are provided for each collector, and the usual decantation process is applied as described in earlier notes on Rand slime practice.

Concentrates Roasted and Cyanided

The concentrates from the vanners, amounting to 8 or 10 tons a day, contain an average of 4½ oz. gold per ton. After settling and draining, this product is treated in a Merton roaster and then ground to slime in Forwood-Down pans. The ground

FIG. 2.—SAND RETREATMENT PLANT, GLOBE AND PHOENIX.

pulp is thickened to a consistency of 1 part solution to 1 part solids, agitated in paddle agitators, and pumped to a Dehne plate-and-frame press. This treatment has proved very efficient, and an extraction of 96 per cent is obtained. Zinc shavings are used for precipitation.

Continuous Decantation for Reground Sand

The extraction made in the sand-leaching department is not good, owing to the presence of antimony in the ore, and the tailings are discharged with an average value of $4 per ton.

This material has accumulated until the dumps contain approximately 250,000 tons of sand, worth say, $1,000,000. It was for the retreatment of this product that the second mill was designed and erected. The plant is interesting, both from a metallurgical point of view, and because it is the first in South Africa to employ continuous counter-current decantation. It is now treating 350 tons of reground sand per day.

A general view of the plant is shown in Fig. 2. It comprises

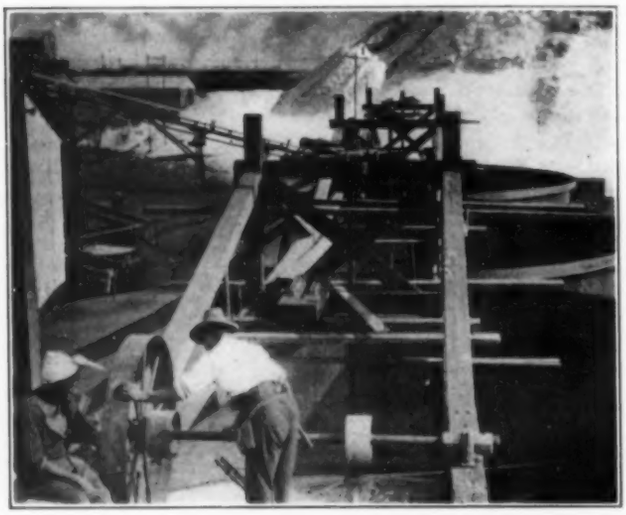


FIG. 3.—THICKENERS AT RETREATMENT PLANT, SAND DUMP AND BELT CONVEYOR IN REAR

three duplex Edwards roasting furnaces, duplex vertical 16-in. plunger pump, spitzkasten, twelve 5-ft. Forwood-Down grinding pans, blanket tables, cones, two 22 ft. by 5 ft. 6 in. tube mills, one 35 ft. by 12 ft. and four 40 ft. by 12 ft. Dorr thickeners, eight 10 ft. by 5 ft. paddle agitators, and centrifugal pumps. In Fig. 2 the roasters, plunger pump and one Dorr thickener are at the right, and four Dorr thickeners at the left. In the center is the structure housing the centrifugal pumps for the continuous decantation system. In the middle foreground are four of the agitators, the other four being behind the building. Fig. 3 gives a closer view of the line of thickeners, and shows also a part of the main sand dump, with inclined belt con-

FIG. 4.—ROASTERS AT GLOBE AND PHOENIX RETREATMENT PLANT

veyor for transporting the sand from the dump to the roaster bins. Fig. 4 gives a view of the roaster buildings, with the conveyor and bins at the right, and plunger pump at the left. A closer view of the paddle agitators is given in the foreground of Fig. 5.

Sand Roasted to Remove Antimony

As already indicated, the sand is removed from the dump and elevated by belt conveyor to an 800-ton steel bin at one end of the roaster building. Push conveyors feed the sand to the roasters, where it is heated to a temperature sufficient to drive off the antimony. An interesting point in connection with the operation of the roasters is that the fuel is coal brought from the Wankie fields near the Victoria Falls on the Zambesi river. Although the coal has to be hauled by rail a very great distance, say 500 miles, it is found cheaper than wood, for the local timber has been removed for many miles around.

The roasted ore is carried by push conveyors to a sump where it is mixed with the solution overflowing the second thickener of the decantation system. This pulp, having a consistency of 5 parts solution to 1 of sand, is raised by the plunger pump to a large spitzkasten. The underflow is ground in pans and passed over blankets to remove gold that is too coarse to be recovered by cyanidation. The overflow from the spitzkasten passes to cones, the underflow of which is ground in tube mills. Finally, all the pulp from pans and tube mills, which is practically minus 150-mesh, gravitates to the pump and again passes through the classifying system.

Thus only the overflow from the two tube-mill cones passes to the continuous decantation system. The pulp is first thickened in the 35-ft. tank, then passes through the agitators to effect solution of the gold, and finally through the 40-ft. thickeners where it receives counter-current washing and decanta-

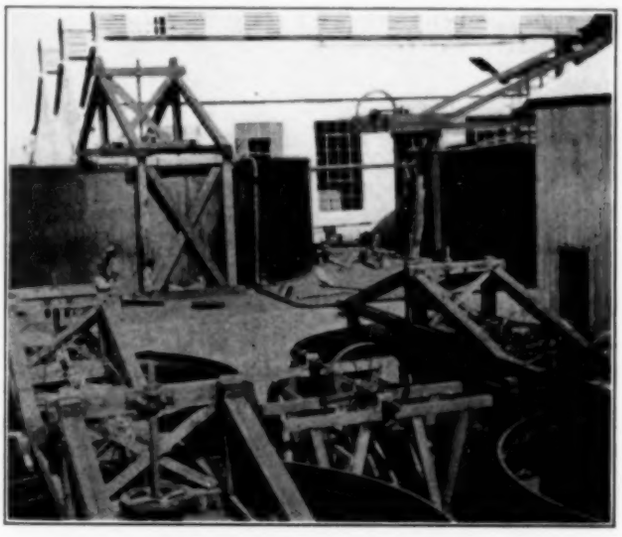


FIG. 5.—PADDLE AGITATORS IN FOREGROUND, GLOBE AND PHOENIX RETREATMENT PLANT

tion according to standard practice as used in similar plants in the United States.[1]

All the thickeners are fitted with spring-alarm devices, which are of considerable advantage in the treatment of material which settles readily and which might accumulate so rapidly as to interfere with the movement of the rakes. The thickened pulp is discharged through 1-in. nozzles. All of the pumping from tank to tank is done by belt-driven centrifugal pumps placed in a central building. An extraction of from 88 per cent to 90 per cent is being obtained in this plant, and the operating costs are very low.

Lonely Reef Mill

About 60 miles north of Bulawayo is a comparatively small mine, the Lonely Reef, which has been a consistent dividend producer for a number of years. A general view of the mill is given in Fig. 6. The equipment comprises 15 stamps, to which 5 are now being added; three tube mills, amalgamating plates, four 20 ft. by 10 ft. Dorr thickeners, four 22 ft. by 8 ft. Pachuca agitators, and four Dehne plate-and-frame presses. Practically all of this equipment is shown plainly in the view of the plant, the stamps being housed in the large building in the background, and the filter presses in the build-

[1]Continuous Decantation has been described in this journal: July, 1911; January, 1912; January, 1913; August, 1913.

ing at the right. A closer view of the thickeners is given in Fig. 7.

The ore is crushed in cyanide solution, through ¼-in. screens, and classified in cones. The cone underflow is reground in tube mills and passed over amalgamating plates. The tubes have silex lining, but hard rock of local origin is used for pebbles. A closed circuit is established between the tube mills and cones, and the pulp is returned to the latter after amalgamation. The cone overflow is very finely ground, and from 90 per cent to 92 per cent will pass a 200-mesh screen. This product is thickened to about 50 per cent moisture, in which condition it is elevated by plunger pumps to the Pachuca tanks which are connected for continuous agitation. The treatment is finished by filter pressing, and the tailing is trammed to the dump as shown in Fig. 6. Presses are used in preference to

FIG. 6.—GENERAL VIEW, LONELY REEF MILL, RHODESIA

other forms of filter on account of their greater efficiency in recovering water.

Excellent metallurgical work is done at this mill. The ore contains an average of 1 oz. gold per ton, and the quantity treated daily is 150 tons. With the addition of five stamps the daily capacity will be proportionately increased.

Concrete Slime Collectors at Bushtick Mill

An interesting feature in connection with operations at the Bushtick mill, is the use of concrete slime vats, 85 ft. diameter and 7 ft. deep. A view of one of these vats is given in Fig. 8, which shows the cracking of the concrete due to the settling of the vat. Various materials have been tried for filling and patching these cracks, but pitch has been found the best.

The ore at the Bushtick is low-grade, averaging about $5 per ton. Twenty ordinary gravity stamps and six Nissen stamps crush in cyanide solution, through screens of ½-in. mesh. The stamp product passes directly to tube mills, and after being reground is run over blankets which retain a rich

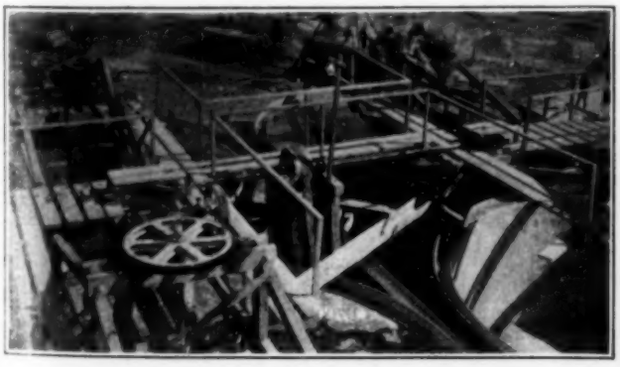


FIG. 7.—SLIME THICKENERS AT LONELY REEF MILL

concentrate representing a value of about $1.50 per ton of ore treated. The tube mills work in closed circuit with pointed-box classifiers, the overflow from which passes to concrete settling pits. Here the sandy portion of the pulp settles and the slime overflows with solution, gravitating to a series of four 85-ft. concrete treatment vats, referred to above. The slime is treated by the ordinary system of intermittent decantation, the pulp being transferred by centrifugal pumps.

All of the power used at this mine is generated by gas engines, direct-connected with electric generators. Gas is produced from charcoal.

A report of operations for the month of May, 1913, shows

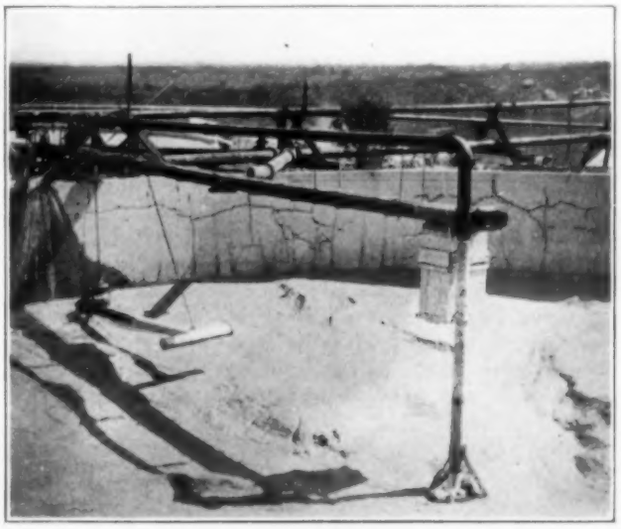


FIG. 8.—85-FT. CONCRETE SLIME COLLECTORS, BUSHTICK MILL

that the mill ran 25 days, crushing 6,106 tons of ore. The total yield was 1,308 oz. fine gold, of which 1,068 oz. was recovered by cyanidation of sand and slime. The duty per stamp per 24 hours was 10.13 tons.

Crushing in Water at the Gaika Mill

There are several features of interest at the Gaika mill. Fifty per cent of the ore is a steatitic rock, very greasy and producing considerable slime. Trommel screens of 1¼-in. mesh precede the stamps. The undersize, amounting to about 30 per cent of the ore treated, by-passes the stamps, and the oversize is crushed in water. The mortar screens also are 1¼-in. mesh, which is the upper limiting size of the feed to Chilean mills. After being reground to 30-mesh in the latter machines, the ore is passed over amalgamating plates and blankets.

A view of the cyanide department at the Gaika is given in Fig. 9, showing the elevated sand-leaching vats at the right, Dorr thickener in the center, and elevated slime storage tanks at the left. Behind the latter is a structure housing the vacuum

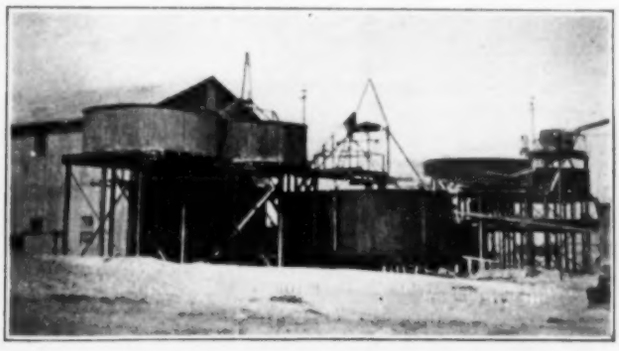


FIG. 9.—SAND AND SLIME DEPARTMENT, SARKA MILL

filtration plant. A closer view of the top of the thickener is given in Fig 10, which is reproduced to show the crank and pulley occasionally used to operate the thickener by hand. It sometimes happens that the power is cut off, in which event the machine is easily operated by hand at normal speed by a Kaffir.

Intermittent Discharge from Thickener

The Chilean mill product is classified in cones, and the sand is treated by leaching. The slime consists of 12 parts water and 1 part solids. It gravitates to the thickener which is discharged intermittently, say once an hour, to provide charges for the filter. Operated in this way the discharged pulp contains only 30 per cent moisture. It is then mixed with cyanide solution and agitated by circulation through a centrifugal pump connected with the filter storage-tank.

The filter was the first of the vacuum type to be used in South Africa, having been built in Australia and transported to Rhodesia. It is of the stationary type and does good work, but owing to the nature of the slime it is impossible to treat cakes of more than ¾ in. thickness.

New Mills Under Construction

As already intimated, Rhodesia has been and still is principally a country of small and rich mines; but the future mining prosperity of the region will depend more on the successful treatment of the larger deposits of low-grade ore. For this purpose several large mills are now being built.

The largest of these is at the Shamva mine, where a mill of

FIG. 10.—THICKENER ADAPTED TO BE OPERATED BY HAND

2,000 tons daily capacity is nearly ready for operation. The ore will be crushed by Nissen stamps in cyanide solution; preliminary classification will be effected in cones, and the sand will be reground in tube mills. After final classification the slime will be thickened in Dorr machines, agitated in Pachuca tanks and filtered in a Butters vacuum filter. Zinc-dust precipitation will be used, according to the Merrill system.

At the Cam and Motor mine a large dry-crushing mill is being built. The ore will be dried, crushed in ball mills and roasted in furnaces of the Edwards type. The roasted ore will be mixed with cyanide solution and classified in cones. The sand will be reground in tube mills, and after reclassification of the pulp, the sand will be leached and the slime filtered by vacuum.

On account of the great scarcity of water in parts of Rhodesia, dry-crushing is likely to find wider application, and several mills of this type are being erected at such mines as the Antelope and Planet-Arcturus.

Flotation of Gold-Copper Ore

A thoroughly modern mill is now being built at the Falcon mines to treat daily about 400 tons of gold-copper ore. After crushing in Nissen stamps, the ore will be concentrated on tables to remove coarse mineral. The tailings will be separated into sand and slime, and the former reground in tube mills. The combined reground sand and slime will then be dewatered and concentrated by flotation, using the Minerals Separation process. This will be the first flotation plant in South Africa, and is suggestive of the recent advances in the concentration of slime and finely ground sand.

A number of other mills, mostly of small capacity, are in course of erection or design. The reader who has followed these notes will readily perceive the modern character of Rhodesian milling plants, and the up-to-date methods of handling slime for cyanidation and flotation, as compared with those in vogue in some other countries. The Rhodesian metallurgist is progressive, and is to be congratulated on results already obtained under adverse conditions, and on the prospect of solving further problems in ore-treatment in that country.

Denver, Colorado.

Carborundum Refractories

By F. J. Tone

Carborundum, while long recognized as a substance of remarkable refractory properties, has never taken an important place among the materials at the disposal of the metallurgist or furnace builder. This has not been due to the material itself but rather to the methods of manufacture into refractory forms.

The first to recognize the refractory properties of carborundum was its discoverer, Acheson, who describes it in his original patent as a substance possessing the properties of great refractibility.

The first patent describing the manufacture of brick and other refractory articles was Acheson No. 615,648, which describes the manufacture of carborundum crucibles, bricks and other forms by the use of highly ferruginous clays and other iron compounds. This was followed by Talbot, with a process relating to the production of neutral furnace linings using a bond of coal tar, by using especially amorphous carborundum. Other early patents were those of Engels, covering the coating of firebrick and furnace walls with carborundum and the manufacture of carborundum muffles; and the patent of Fitz Gerald, who invented a process of recrystallizing highly refractory and solid articles without any foreign binder.

Two types of carborundum refractories are known; one containing a vitrified or ceramic binder, the other consisting entirely of carborundum. This latter type is made according to the Tone patents, Nos. 992,698 and 1,013,700, which in general cover the silicidizing of carbon forms and their conversion into silicon carbide by subjecting the carbon to silicon-containing vapors at a high temperature. Patent No. 992,698 describes a preferred method in which silicon carbide grains are molded with an addition of carbon. When these bricks are heated in a mixture which gives off silicon-containing vapors, as, for example, in the ordinary carborundum furnace, the carbon is converted into carborundum, forming a dense, interstitial binder or matrix for the silicon carbide grains.

U. S. Patent No. 1,013,700 covers the conversion of pure carbon forms into carborundum, the resulting material being in all respects equivalent to the silundum described in Bölling's German patents. The latter process is applicable only to those forms of small cross sections. Silicon vapors cannot, without great difficulty, penetrate a pure carbon article to a depth exceeding ¼ in.

This type of refractory is absolutely unattacked by acids and is inert to the influence of practically all other chemical agents. It is, however, extremely expensive to produce and its use is limited to those places where the ordinary brand of carborundum refractory is not suitable. Thus for certain parts

of electric furnaces, as, for example, for heating furnaces of the carborundum-resistance type, it is the only material suitable for placing in contact with the carbon resistor. It also has advantages in constructing electric furnaces of the Hoskins carbon-plate type, where, owing to the resistance to oxidation of the silicon carbide coating, the life of the carbon plates is increased five to ten times. It also has remarkable value for the construction of roofs of electric steel furnaces and furnaces of a similar type.

The vitrified carborundum refractory has, owing to its lower cost of production, a much wider application to the refractory industry. It consists of suitably chosen carborundum grains bonded with a special form of refractory clay, mixtures of grits and methods of molding being employed to insure a very dense body of low porosity. The brick are fired in kilns of a new design, giving a range of temperatures never before attained in commercial kilns used for the production of refractories. The clay binder is completely vitrified so that only a small quantity is necessary to bind the highly compacted mass.

Carborundum is a material unique among refractories due to the following distinguishing physical properties:

(1) High softening or decomposition point.
(2) High thermal conductivity.
(3) Low coefficient of expansion.
(4) Hardness and mechanical strength.
(5) Resistance to acids and other chemical agents.
(6) Comparatively low density.

High Softening or Decomposition Point

According to the most recent determinations crystalline carborundum forms at 1,840 deg. C. and dissociates into its elements at 2,240 deg. C., the silicon volatizing off as a vapor, the carbon remaining as graphite. No fusion or softening occurs below the dissociation temperature, as is shown by the sharp and perfect forms of graphite pseudomorphs or "skeletons" left when carborundum dissociates.

Carborundum refractories retain their high mechanical strength practically intact at high temperatures, which distinguishes it from other refractories since most refractories undergo deformation especially when subjected to mechanical stress at high temperatures. Thus, for example, silica brick, chrome brick and fireclay brick soften at a temperature several hundred degrees lower than their fusion point, which is not the case with carborundum refractories.

When bonded with a clay binder carborundum brick have very great refractory value. However, it is recognized now that the ideal condition to be obtained is that the amount of binding material used be the least consistent with the requisite mechanical strength and that the vitrification temperature of the binder be as high as is commercially practicable.

Failure to realize these essential conditions has been the cause for the indifferent success of carborundum refractories where their use first appeared most promising. In general the refractoriness of a conglomerate refractory mass is a direct function of the amount and character of the least refractory constituent, and the recognition of these facts is what now permits the manufacture of carborundum refractories which are giving such satisfactory results.

Thermal Conductivity and Capacity

Carborundum refractories have a thermal conductivity five or six times that of fireclay. Wolgodine has determined the conductivity of various refractories expressed in gram calories per second per cubic centimeter per 1 deg. C.

Carborundum brick	0.0231
Magnesia brick	0.0071
Chrome brick	0.0057
Fireclay brick	0.0042
Silica brick	0.0020

We have been accustomed to regard heat insulation as a property in firebrick equally desirable with resistance to fusion, but a close study of furnace work reveals numerous locations where heat conductivity is the chief desideratum. These are muffle furnaces and indirect-heating furnaces of all types, such as zinc, retorts, partition walls in by-product coke ovens, pottery saggers, bats, etc., kiln floors, recuperators, regenerators, etc.

The transmission of heat depends on the diffusivity which is given by the relation

$$D = \frac{K}{W S}$$

where D = diffusity.
K = heat conductivity in gram calories per cubic centimeter per 1 deg. C.
W = density.
S = specific heat.

Based on a temperature of 1000 deg. C., the value of D for carborundum refractories is 0.0578, whereas for firebrick it varies from 0.0067 to 0.0094. That is, carborundum brick show an efficiency of six to nine times that of firebrick in the transmission of heat. Further, for a given time and for a given temperature of the atmosphere of furnaces more heat will be absorbed and a higher temperature reached than with fireclay.

The specific heat of fireclay brick is 0.2243, that of carborundum brick about 0.19. The density of a corborundum brick is about that of fireclay. The heat capacity is less and consequently the absorbtion and transmission of heat is correspondingly greater.

Low Coefficient of Expansion

Carborundum has a coefficient of expansion of 0.0000047 per degree from 200 to 900 deg. C. That of firebrick varies from 0.00000567 to 0.0000161; that of fused alumina is 0.00000866. Carborundum brick are practically indestructible from changes in temperature. White-hot bricks can be thrown into water without breaking. This property is made use of in building furnace structures where leakage, due to alternate contraction and expansion is an important factor. This indestructibility of carborundum refractories under differential heat treatment is also aided by the extremely high thermal conductivity which aids in dissipating the heat.

Hardness and Mechanical Strength

Since carborundum refractories have almost no coefficient of expansion, no porous structure is necessary and a high density is easily obtained. A porosity of 15-20 per cent is obtained against 25-30 per cent for ordinary fireclay brick. This results in extraordinary mechanical strength and combined with the great hardness of carborundum which is 9.6 on Moh's scale of hardness (diamond 10, corundum 9) gives the material a remarkable resistance to abrasion. Locations where this property is of paramount importance are the floors of coke ovens subject to the wear of charging and discharging apparatus, the bottoms of Lehr furnaces for making glass, the floors of heating furnaces and furnace parts subject to the attrition of rapidly moving gases.

Another factor influencing the mechanical strength at high temperature is the moleculer rearrangements at certain critical temperatures. Tests show that magnesia refractories may fail at 1550 deg. C., chrome brick at 1450 deg. C., from this cause. Silica changes into tridymite at about 1200 deg. C. and is consequently subject to deformation from this cause.

Carborundum is not subject to any molecular changes of this character having been formed at 1820 deg C. in a crystalline state and having no allotropic modifications.

Resistance to Chemical Action

Carborundum is absolutely unacted upon by any acid when in the liquid or gaseous state; the only active acid reagent being a mixture of hydrofluoric and nitric acids. It is, however, decomposed by alkali, but only when subjected to a high temperature. When used in metallurgical structures it develops a brownish maroon glaze which prevents any subsequent action of slags, fused ashes, etc., and also practically prevents

any oxidation taking place. This glaze results from the interaction of the carborundum and the binder and as it is developed in the body of the refractory is extremely tenacious and impervious.

Density

Carborundum has a specific gravity of 3.1 to 3.2; when made up into refractory form its density is 2.1 to 2.5. This comparatively low density lessens the mechanical stress developed at high temperatures owing to the small weight of the refractory structures.

The Carborundum Company, Niagara Falls, N. Y., has placed on the market the two brands of refractories mentioned above. The name "Carbofrax" brand has been given to that formed of carborundum with a citrified binder and the name "Refrax" brand to that formed of carborundum entirely. Both varieties can be furnished in any regular or special forms, as regular, arch and key, brick, retorts, crucibles, base blocks, muffles, tubes, etc.

Hydrometallurgy

Joys of Its Theory, Woes of Its Practice

By Regis Chauvenet

Hydrometallurgy is a broad term if it be held to include every process in which the metal to be won is, at any stage of the operation, in solution.

The present article does not deal with any one process, whether successful or the reverse, nor shall we attempt to describe operative details, further than to indicate their general scope or chemical significance. We wish to point out why some well-heralded processes are relegated, not exactly as "have-beens" but rather as "never-weres," and why it is to be feared a similar fate awaits more than one attempt now in the course of evolution, i. e., in the hopes of its stockholders.

The difficulties met with in solution processes are many. For the most part they are not based upon theoretical error. The troubles encountered, a few of which we shall present, are partly due to manipulatory obstacles, partly to underestimates of expense. We cannot specify individual cases, whether prospective or operative, for obvious reasons. Nor are the difficulties to be described peculiar to any one process, or even to any allied group. We hope to show in what directions miscalculations have been common, as a warning both to over-enthusiastic inventors and to prospective investors.

Every metallurgical process involves the operation of chemical laws. This may be generally recognized today; possibly not always in times past.

"In Spite of Science"

In a by-gone day the writer, then in St. Louis, Mo., was visited by an official of the Missouri Tin Company, a corporation formed for the purpose of extracting tin from porphyry. In justice to the promoters, they did not so state it in their prospectus. The functionary aforesaid, after much earnest protest against the assertion that there was no tin in the prophyry, a statement for which the writer was responsible, wound up with the following amazing dictum: *"Why, we know as well as you do that the tin in our ore isn't susceptible of chemical detection."*

It was almost prophetic—for did not the renowned "Metallurgical Murphy" years later, issue his immortal report: "The ore is of metallic contents, but basic constituents. Its values, while susceptible of chemical detection, are not capable of metallurgical extraction!"

Chemical Ignorance

We have said that difficulties encountered in actual practice were for the most part not based upon theoretical error. However, this is hardly true of many processes which are still in the purely prospective stage.

By "ignorance" of chemistry we do not mean total ignorance of text-book knowledge of that subject. It is a frequent happening, however, that the inventor of a chemical process knows just enough chemistry to harm him. He has read of, possibly practiced, certain reactions, and tries them under conditions wholly favorable to their success. Of reactions in complex solutions, or the influence of mass; of the solvent powers of his own solutions for supposedly "insoluble" substances—of a hundred other complications, familiar to all operative chemists, he knows little or nothing. We divide the style of ignorance too commonly found in attempted chemical metallurgy, into "scientific, manipulatory and financial." In the following examples of the "left-wing" cases (i. e., frauds pure and simple), we divide our criticism between the fakir and the dupe; for the latter ordinarily has ample means and opportunity to discover in advance the false or fraudulent grounds upon which he is asked to invest.

Frauds, Fakirs and Fables

In the above classification we have not included the ignoramus "in toto." We nevertheless shall give one frightful example of him. King Solomon has disposed of him in a well-known phrase, which we may put into modern technical language by saying that one may pound him to forty mesh in a ten-stamp mill without changing his mental characteristics. Most metallurgical men will recall without difficulty several cases of attempts at the impossible. We have at the outset to dispose of the ever-recurrent metallurgical fraud—fool he may be, fraud he certainly is—whose mission is to create matter, or in brief, to extract metal, not from where it is, but from where it isn't!

"Ven ve are dead," said poor old Rip van Winkle, "how soon ve are forgotten!" And these recurrent spasms of alchemistic foolery, which appear

"Each in their turn, to make the vulgar stare,"

how soon they are forgotten in this busy world. Yet, within the memory of thousands of living men, we have had three schemes of continental fame, and half a dozen of more local celebrity, all claiming the production of gold, some under one, some under another disguise, as to method; but all in fact, if not in form, claiming the rediscovery of the art of transmutation.

(Strange, by the way, that all of these "transmutors" or what-ever-you-may-call-em's, never discovered any transmutation except into gold. Possibly one or two of them may have included silver.)

For the strictly technical reader, it may be a waste of time to read over accounts of mere trickery. Nevertheless some of these now historical transactions are reproduced. "To what end?" may be asked. Well, as a very slight contribution to the science of psychology. There is no appeal to science in these wornout juggler's tricks, but if we venture to classify them, as we think we can, and show that they fall under certain definite forms of appeal to ignorance, prejudice, credulity or greed, we may possibly indicate to some readers the ear marks by which he may recognize a fraud, without having too closely to investigate the alleged technical details.

"Classified Ignorance"

These hydrometallurgical attempts then, in so far as they try to get the general public interested, are divided into:

(1.) Appeals to total ignorance.

(2.) Appeals to anti-scientific prejudice.

(3.) Assumed appeals to scientific principles, under a further assumption that the inventor (and he alone) has learned how to apply these profound truths to practical ends.

First: Appeal to sheer ignorance. The best example that occurs to us is that of the "Wynne" process, started something over a decade ago in Denver, Colo. The promoters so far succeeded that they actually sold stock in the process to no small amount. The "recipe" was kept secret, but the results were apparent. Prospectors and mine owners were invited to send in samples of their ores, and did so. Invariably the "return" was made showing values far in excess of report made by any other assayer. But the new concern went further than this: in many instances it actually turned back to the sender of the ore sample the gold "extracted."

Not the "ignorant class" if you please, but some men of prominence and of presumable education were taken in by this particularly stupid trick.

"Of course," you will say, "there was some attempted explanation of the phenomenon?" No. Not so far as the public who were invited to invest were concerned. On its naked, self-asserted production of gold from rock, barren or nearly barren, the delusion found many believers. It would seem that the return of the metal in a few selected cases was productive of "results."

The "inventor" died. His stockholders were heirs to his chemical "recipe." They tried it, reporting, with refreshing naïveté, that it wouldn't work." No wonder. It is before us now, a silly mess of potash, nitric acid, ammonia, lithium carbonate and other ingredients. Possibly a fair bug poison, but fit for nothing else that we can imagine. The stockholders pocketed their loss and the public forgot about it in a fortnight.

Second: Appeal to anti-scientific prejudice. We should like to annex the proper name to this little bit of local history, but we regard the "feelin's" of some possible readers, who may still be a little sore. We shall call the camp by what it is in fact, not by its map name. Busted Camp, then, had "ore" and had also many enthusiastic promoters. The "formations" were identical with those of Cripple Creek. (Of course, for the discovery was made in the days of first development of the wonderful riches of the latter famous camp.) Everything was lovely at Busted, except that there was no gold in the ore. However, the settlers were true "Boosters" and weren't going to let a little thing like that discourage them. They invented a new assay method, which they called a "color method," but we shall not try our reader's patience with a description of the fool thing. Suffice it that the great argument used was based upon the fact that no "scientific" man or "regular" assayer or chemist, could find the gold here. *That*, it seems was the main point of value!

For, there is no doubt about it, there is an element in every community who have deep-seated convictions that "science" is a fraud. They could not define "science" to save their dollars—but as a noted foreign agitator said as he landed in New York, "whatever your government is, I'm against it," so there are certainly those who say "whatever science says about this, I don't believe it!" And of these were the investors in Busted Camp: the "poor but honest" and later, bankrupt people, who invested in their own ignorance—and lost.

The perpetrators of this fraud knew, of course, what they were doing. They also knew how to calculate upon ignorance and prejudice. That is the characteristic of fraudulent promoters everywhere. They are shrewd enough to gauge the prejudices of those with whom they deal. They descend to their level in their solicitations. Finding people who were already prepared to believe in a thing *in the precise ratio in which it had no scientific bearing*, they play upon that particular form of ignorance, and by skillful sneers at "scientific" work, induce the belief that work which is *not* "scientific," must be correct!

Is the logic apparent? It was at least successful, and many a "share" was sold. It was another proof of the prevalence of scientific illiteracy. We may roughly state the proportion of scientific illiterates at ninety per cent of the population. The only connection of this with "hydrometallurgy" is found in the fact that the assay method by which they found the gold that wasn't there, was a "wet" method, and that the proposed method of extraction was also "hydrometallurgical."

The Ocean a Gold Mine

Third in our classification comes the assumption of scientific principles. The "Electrolytic Salts Company" had a wonderful history. Mr. Jernegan issued circulars stating among other things that it was "well known" that sea water contained gold. The whole statement was a mixture of falsehood and of semi-truth. The object of the enterprise was not to extract sunbeams from cucumbers, but gold from the ocean.

It is indeed a "well-known" fact that for many years a paragraph has been floating around the world stating that sea water contains a definite proportion of gold. One metallurgist has taken the trouble to trace the statement to its source, with the result that the trail "ran up a tree." No authoritative statement of the contents of gold "per ton of ocean" could be discovered. But it was enough to start the remarkable exploitation now briefly to be described.

The inventor stated the amount of gold to be one grain per ton of sea water. Possibly there is some infinitesimal trace of gold there. Certainly it is not that much—but the amount matters very little. The ingenuity of the attack (on the public) consisted in the declaration that the inventor was well aware that he couldn't handle a ton of material for four cents, which is about the value of one grain of gold. He added however, that if we could persuade "Nature" to do the job for us, that little difficulty would be overcome.

Nature has so shaped the Bay of Fundy that tides of extraordinary altitude characterize its upper reaches. "Here we shall place our mill," said the inventor.

The tanks or reservoirs to hold the sea water are to be so placed that they will fill at high tide, and empty themselves at low tide. Here is Nature's handling of the material. But how about the extraction of the gold? Why, that was so simple that "it was to smile."

Here we have a gigantic battery of one fluid (the sea water) and one cell (the reservoir). There were, of course, "plates" for the deposition of the gold. In flows the water, down goes the gold. The only "work" is to scrape off the beautiful stuff—and there you are.

It was indeed a lovely scheme. The hokus of the original statement aided by the pocus of the "battery" needed only the locus (Bay of Fundy) to bring it to a focus (subscriptions and stockholders). And the subscriptions flowed in; the mill was actually built. The scoffers scoffed, of course, but suddenly they were put to shame. Lo, a dividend! A real, fifteen-thousand-dollar dividend, paid in hard cash. With it, perhaps the most ingenious statement of all, viz.: "The yield was a great disappointment, being but one-fourth of the total gold content. From this trivial portion the dividend has been declared. But the cause of the small yield has been discovered, and the matter is easily rectified," and so on.

Now indeed, money came in fast. There had been just enough apparent appeal to scientific principles to satisfy those to whom electrolytic operations are a fact sufficiently well known, but who have never bothered themselves as to any details. Doubts were dispelled by the money. The stale old trick of a dividend paid from stock subscriptions had worked once more.

Then came the closing of the mill, the flight of the promoter to a more congenial climate, and other highly uninteresting details. It is said that the fugitive was caught abroad, but we are not able to give the closing chapters of this miserable history.

Enough of these flights of metallurgical fancy which are, pardon a mixed simile, but sample bricks (gold bricks) from many an edifice of fraud, testifying at once to the credulity and scientific ignorance of a supposedly enlightened public. It was not to discuss such baseless frivolities that we started to pen this brief essay. There are plenty of cases outside of mere trickery, and to some of these we now turn our attention. Many processes are predestined to failure whose theory is fair, and whose intent is honest.

Trouble Begins at the Mine

We wholly omit the cyanide process. It constitutes a class by itself. This being understood, we must add to save misapprehension, that we do not undertake to criticize any one process. Our object is to point out certain difficulties inherent in most operations in which "wet" methods are in use, characterizing no single case, that is, no one as apart from most of the others.

Most "wet" methods may be said to be attempts to repeat on a large scale the operations of the analytical laboratory. Remotely, they exhibit, in their experimental stage, the same

relation to the commercial plant that a model does to the actual machine.

It is a matter of common remark among mechanical engineers that the perfect working of a model by no means insures the successful operation of the "real thing." We may say the same regarding the working of a chemical process, albeit the reasons are not quite the same for failure in the latter case.

The fallacy is not always apparent at first glance. There is usually an experimental plant in miniature, in which everything works nicely, for we are now assuming that the prospective working plant is based upon actual results obtained in a small way. This again, implies that the chemical principles are correct. The reasoning which carries us to the larger operation is exceedingly simple. Here is a result obtained with one ounce—no matter of what. Multiply by thirty-two thousand, and we have the outcome of the same operation on one ton. But we are seeking the erection of that favorite unit of the metallurgical inventor, a "hundred-ton mill." Very well. Multiply the last figure by one hundred—and there you are!

In this little model plant, solutions are made in vessels quite impervious to the action of chemicals, whether solvents or precipitants. Precipitation and filtration proceed rapidly and regularly. When you need a hotter solution, all you have to do is to turn on a gas jet, the cost of which (on a few liters of solution) is too trivial to take note of. Your little battery, too, how small a bother it is. Now and then an anode may wear down, but you have a box of new ones handy.

An unfortunate necessity of most hydrometallurgical processes is uniformity in the composition and tenor of the ore treated. Hence our caption, "Trouble begins at the mine."

In smelting ores, there may be a large margin of variation, within which it is no great trouble or expense to change treatment according to the laboratory reports. Take lead smelting. The furnace has been running on ore with little or no zinc. That metal now appears in something more than the trace or the very low percentage which has been the rule. The difficulty is met on the charging floor, with a slag calculation regulating the charge: an increase in the iron, a decrease in the lime, probably an allowance for less total silica. The intractable element is fluxed off, not quite as easily as the ordinary constituents, but certainly without visible interruption of current operations.

Not so in our "hydro" scheme. We may be getting our metals into solution by direct action, or possibly by previous chemical preparation (ex. gr. chloridization), but when a new constituent, or a marked change in proportions of old ones comes along, the changes involved are complex. Later we shall return to this point in another form, showing the necessity of too much "expert" work in "scientific" plants.

The differences in treatment due to change in composition, form a theme of too much detail to be taken up here. All we want is to present such changes as one of the "Woes" of hydrometallurgy. A great custom smelter can throw all sorts on its bedding-floor, and come out with a fair average. Rare indeed would be the instances in which this could be done by a wet-treatment works.

One class of material for wet treatment consists of ore too low for smelting and (if gold) not fit for amalgamation. The first question arising is whether the raw material can be directly treated or whether some roast or other preparation should precede. If the latter, we have our first trouble, before the method comes into play at all, for the roasting of low-grade ore means the employment of most of the fuel for keeping the big barren gangue heated.

Whether roasting has or has not preceded solution, we have as difficulty number two, the repetition in another form of the fuel waste involved in roasting a low-grade ore. Treatment of the ore with any solvent means the application of that solvent to all of the barren gangue as well as to the ore proper. This brings in a question to be met more in detail a little later, viz., the trouble arising from too great a dilution of solutions.

The difference between large and small operations in the analytical methods of ore treatment are barely indicated in the above preliminary remarks. In fact, the trouble recurs throughout, and we cannot stop at every step to call attention to it. Like the poor, it is "always with us."

"But cannot we concentrate the ore before treatment?" This is a suggestion often heard, and a favorite one with promoters of wet methods. Let us look at it for a moment.

These initial difficulties are not really overcome by concentration. It is true that so far as cost of extraction per unit of metal is concerned, the concentrate is far ahead of the raw ore. Suppose a ratio of twenty into one. We have met with more than one enthusiastic inventor who would figure on his extraction cost on a concentrate, but who became very shy when mining cost was inquired into. But the fact that ninety-five tons had been mined for the five he had treated stood there all the same, like a giant with a big club, guarding the very entrance to the process.

So the operating expense killed the treatment of the raw ore, and the mining expense killed the concentration proposition. Our critic, standing at our elbow remarks, "sarcastic like," that the mining expense kills it in either case. True. But we are trying to be kind to the process, and grant that if it worked on the concentrate, "it was the mine's fault" if the outcome was bad.

Of the many preparations of minerals for solution, we select chloridization, not because it has been much practiced but because certain operations following the initial treatment illustrate the difficulties of chemical treatment in several necessary proceedings. "Chloridization" by the way is a somewhat new term. As it sounds affected, as against the simpler "chlorination," let us here explain that we shall use it to mean the treatment of the ore by "dry chlorine," a very different thing, chemically, from "chlorination" of a wet pulp.

Chloridization has been tried in England, Germany, Australia and the United States. That is, it has been experimented with, though it can hardly be said to have come to a commercial basis. That fact does not exclude it from either scientific or practical investigation. We are not greatly concerned just now with the commercial failures or success, we wish to show the nature of the obstacles which will have to be encountered.

Finally, we wish to emphasize, in order not to be misapprehended, that the difficulties in chemical manipulation on the large scale which we shall partially detail, *are not presented as peculiar to the chloridization process.* They inhere in any scheme in which it is sought to separate metals from one another starting with a solution containing all of them; in short as already indicated, any scheme which approaches in its details, the analytical method for complete separation of all the constituents of a substance under analysis.

The operation starts by treating the ore (usually in a revolving cylinder) with dry chlorine. This probably has been tried out more in England than in any other country. In the United States we have two plants in the experimental stage, one at Helena, Mont., the other at Georgetown, Colo.; there may be others, but we have not heard of them. Although a little aside from the scope of this article, we may note that several methods of getting the chlorine have been proposed The most obvious is by electrolysis from common salt, $NaCl = Na + Cl$. The sodium, however, in the presence of water becomes caustic soda ($NaOH$), and hydrogen is liberated. The reason for mentioning this part of the operation is to remark that it makes quite a difference in the net cost of the product, whether the current at the plant is cheap or dear, also whether there is or is not a ready market for the caustic soda. If there is not, can it be made available in the precipitation of some element in another part of the scheme?

Cost of chlorine produced in this way, is always a somewhat uncertain figure. "One cent" is a common claim; it may be doubted whether this has ever been realized. Two cents a pound is nearer to actual experience. If the caustic soda is used in furthering the process, its value (if it is to be subtracted from the gross cost of chlorine production), is an uncertain factor, greatly depending, we should say, upon the point of view of the promoter.

Another source of chlorine has been zinc chloride, the zinc of course to be regained ultimately, or indeed, directly from the original electrolysis. The metallic zinc thus obtained is again put into solution, being used for the precipitation of metallic lead, i. e., $PbCl_2 + Zn = ZnCl_2 + Pb$. This would be a queer proceeding, economically, were it not for the fact that the relative weights of lead and zinc are at 207 to 65.

Not to anticipate however, let us consider some of the reactions following solution of the chlorides after the reaction in the cylinder is complete. Take the proposed sequence in an early attempt at Helena. It included at one point the complete separation of the iron, as ferric hydroxide. The reagent to be employed was nothing more than the caustic soda made on the spot, and tapped daily from the batteries which evolved the chlorine. Of course, the chemistry is quite correct. Now as to handling the precipitate. We have a hundred-ton mill, and our ore contains let us say 10 per cent of iron. Who shall face the handling of this terrific precipitate? The ten tons of iron make nearly twenty tons of the hydroxide. Have you ever precipitated one gram of metallic iron as ferric hydroxide? It needs about a liter (say a quart) of water for its treatment. Allow for filtration *and washing* a total of only half as much solution as indicated above, and we would have a daily handling of six hundred thousand gallons of water, and cold water would not suffice. For decently quick work in precipitation and especially in filtration we need hot solutions. No wonder that the genius of more than one inventor has been at work to modify or remove this bugbear. It has in fact been met in a later process at Helena, and a far better procedure adopted. Elsewhere, we are told, this hydroxide precipitation is still proposed albeit not in this connection.

Inevitable Losses

Going backward for a moment in the operation, it must be remembered that at every stage we shall have losses. These are not necessarily due to careless operations. Take the original solution of mass from the cylinder. Let us suppose that we have lead, zinc and iron as main constituents, and 50 per cent gangue. Lead is present as chloride. Wash it out. You may be pretty sure of this, that when the lead is washed out all the other chlorides will be also. (We are neglecting silver for the moment.) But this washing uses hot water, as $PbCl_2$ is insoluble in cold water.

In the Helena process, heat is applied long before this, i.e., to the cylinder itself, in order to promote the decomposition of sulphur chlorides.

The absolute necessity of heat at various stages of solution processes is an expense much overlooked. Few who have not tried it are aware of the enormous gain in time and clean work, in precipitating and washing, of hot solutions over cold. Yet water persists in holding on to its high specific heat, and few low-grade processes are there but that will go down, economically, *under this expense alone*.

But those losses? When we have washed out a total of 90 per cent of assay value we have done very well. For it will not do to assume that our metals are wholly converted to soluble forms. Again when we have precipitated and filtered a big tonnage of ferric hydroxide, we shall be doing not badly if our filtrate contains 95 per cent of theory, for we cannot wash as would an analytical chemist. We shall have other precipitations. At every one, another deficit. After two operations (in addition to original solution), we may figure something after this fashion, "Ninety per cent of ninety-five per cent and ninety-five per cent of the latter weight." That would be about eighty-one per cent. These successive deficits in work of this description are to be overcome only by very clean and hot precipitations, and enormous amounts of wash water. But the latter brings about a new source of expense, not alone in heating the water, but in the unbearable dilution. Tanks upon tanks and then more tanks. There is hardly an operation of the kind that is not "up against" some such questions all the time, ex. gr., whether to leave a constituent behind, losing it, or wash it out at an ultimate expense greater than its value, when all of the extra installation, time and handling is accounted for.

We said above that much ingenuity had been called into play to avoid the necessity of handling the enormous iron precipitate necessitated in this and other similar operations. We shall pass over this rather quickly.

As to methods for disposing of this great bulk, one is simple enough in principle, being "not to produce it at all." This is effected in the Malm process by roasting out the chlorine from the iron chloride, thus regaining chlorine which is simultaneously used for attacking fresh ore. The iron oxide now left behind is insoluble. As an economic procedure this shows one striking advantage, for if we precipitate hydroxide with caustic soda, we get the chlorine back in its original form of sodium chloride, that is we have obtained an expensive reagent combined in the form from which it came.

Another method is to use as precipitant a substance which yields a much less bulky product than by caustic soda. Zinc oxide will do this. It has also been proposed in this connection to use zinc ore direct, calcined carbonate or even silicate[1]. The topic is an enticing one, but we decline to be tempted aside by it, further than to say that the last named proposition seems to offer a new set of difficulties, we hope not too serious for economic outcome.

This case is presented merely as a type. Iron hydroxide is not the only precipitate which may have to be dealt with. If we have shown that precipitation, filtration and washing, the commonest of operations in analysis, are by no means such simple matters as often assumed in treatment of heavy tonnage, we have accomplished one object of our essay.

As for the general position that the methods of the analytical laboratory are not fitted for large operations, so that arguments fit for grammes are very poor ones as applied to tons, we have already mentioned it. We warn any one who may be invited to aid in projects of this general nature to look to this feature. The argument against a solution process which is based upon no other grounds is insufficient, *taken by itself*.

Having then indicated (1) the difficulties encountered in treating precipitates, (2) the necessity of applying heat at many stages, (3) the losses to be figured on at each stage, we may pass on. In dropping these points we may add one more little note of warning, viz: nearly every inventor figures upon a higher final product than experience warrants. There is a very familiar old argument, still used and still doubtless to be used for many a year, against shipping ore to a smelter. The argument reduced to its fundamental basis is about this: "Do not pay other people for the treatment, but treat your own ore. If they can pay for the ore and make a profit out of the treatment, that last is money that should be in *your* pocket, not in theirs." Strange that this argument should have been many times so misapplied that certain mining enterprises have actually lost money for years on it, yet we believe it to be demonstrable that such has been the case. The general (i.e., the apparent financial) argument has overcome the really technical one. The two points for comparison are, (1) What will treatment per ton cost you, (2) What will be your percentage of production, compared with assay value?

Then as to the "other fellow," who offers to treat your ore for you, the same questions with hardly any variation, viz: (1) What will he charge you per ton for treatment, (2) What will be your actual yield?

Stripped of all the details of freight and other charges this is what it comes to. Aside from treatment-cost look at the element so often lost sight of, the fact that smelting regains the highest percentage of any process, and then make a very simple calculation, the object of which is to find out whether after all it doesn't "cost more than it comes to."

Some may challenge the statement that smelting shows the highest yield of any process. Of course we mean that for a statement holding true of average cases and average conditions. That there are ores so conditioned by constitution, grade and locality that smelting is impossible, is well known. But we know that ores have been locally treated for years which might

[1] It is proposed to use this scheme at Helena.

have been shipped for custom treatment at better profit.

A few chemical difficulties remain to be disposed of. One is the troublesome "entertaining" of an element of value in the precipitation of another. This may be avoided in the analytical laboratory to a great extent. The means used is usually dilution. Hot solutions as a rule are less liable to this "trick." It cannot be lost sight of, being indeed only a special form of the "losses" already mentioned.

Incident to the question of the handling and washing of great masses of material, comes the ever present question of the treatment of very dilute solutions. Here again nothing can be settled except under discussion of the special conditions. There may be two reasons for thorough washing of a precipitate, one being to secure a pure product, the other the presence of an element too valuable to waste in the washings. Both of these reasons may apply in a given case. Nice figuring is called for, or it may be necessary to appeal to actual experience to decide when a solution is past the point of profitable treatment and may be better sent "down the creek" than further bothered with.

Continuous Operation an Essential

A very broad question (and difficulty) arises in the sequence of operations. It is assumed in any operation involving several steps, that each and every one delivers its daily or hourly quantity to the next one in order. The original treatment sends along its solution for precipitation let us say, the second one delivers its precipitate on the one side and its solution on the other, each for further treatment. Here however, occurs some stoppage. No matter what the nature of the hindrance, the general happening is now this, viz: all operations "precedent" come to a stand in a very few hours, i. e., until the obstruction is removed. We do not pretend that such troubles are peculiar to hydrometallurgy, but we believe that they cost more here than anywhere else.

A stoppage of this kind of which we were a witness arose from the sudden discovery that a certain element was present in solution which had no business there. It was small, it was of no value, but its presence would certainly give an "unmerchantable" color to the next product which happened to be zinc, in one or another form according to demand—oxide, carbonate, chloride. How did the objectionable element get there? Here is another of the small miseries of detail. Possibly the solution, containing many reagents and compounds derived from the ore itself, constituted a solvent for the troublesome intruder, which should have been (but wasn't) wholly included in a previous precipitation. No matter how—no matter why. It was impossible to send along a "dirty" solution. The processes ahead stopped for want of material. The processes behind stopped for lack of space to work in. The "experts" (we shall pay our compliments to them presently) have to quickly determine how best to rectify the mistake.

Batteries, Efficiency and Losses

This is delicate ground, on which the professional electrician may challenge almost any statement. We shall not assert that the cost of a given current is commonly underestimated, but when it comes to the annual round-up, which must of course include wear, tear and repair, leakage (both the "juice" and occasionally the chemical solutions), and net chemical product—in short net efficiency, the results are often disheartening.

Many loose assertions are afloat regarding electrolytic decomposition. Take a simple case, based on the original law of Faraday, copper chloride versus cuprous chloride ($CuCl_2$ and Cu_2Cl_2). "The same current that decomposes the one decomposes the other, so that if we reduce the copper solution to cuprous condition we get twice as much metal deposited for the same current." Certainly. The theory is perfect. The "kation" is copper, the "anion" is chlorine. And the kation is twice as weighty in the cuprous salt. You would indeed get twice as much copper for the same current, but for the fact that you don't. You will do very well at 70 per cent increase instead of 100.

Then there is a thing called resistance. Take the very case we started with, decomposition of common salt. Into the theory of its progressive decomposition we do not even attempt to enter. It is well known that as the operation proceeds the current required for a given net evolution of chlorine (and corresponding formation of caustic soda) increases for each unit produced.

For every such decomposition there will be a practical limit, determined for any given works by experience alone. In the case under consideration it may be somewhere between 60 per cent (of total decomposition) and 85. Besides the battery question we have here the additional one of the very impure caustic we have to use, i. e., if it is to be used in subsequent operations.

Incidentally, we have never seen this feature squarely and fairly considered in pre-calculations, i. e., when made by the projectors. There is also a sort of scheme always indicated—perhaps a little dimly—to regain all of this salt at the "final end" of the process, i. e., when the last of the valuable metals has been extracted. As a rule, however, these final dilute salt solutions will "go down the creek."

How Many Experts to the Ton?

No organization of a works running on chemical principles but that would include the scheme of chemists—analytical, operative and supervisory, in its employ. But how many? Most schemes include "wages" in their estimates. There is more call and more constant call for the skilled part of the work in hydrometallurgy than in any other line we can think of. The laboratory force would be at work unceasingly, as a matter of course. But is that all? Far from it. Ordinary labor cannot be depended on in such operations. There is "hurry work" from the time a defect is discovered to the time it is rectified, and, as indicated above, a stoppage of many operations while the remedy is sought and applied. In fact a chemist on each and every landing of the mill would not be amiss. We need an "expert" to detect the trouble, another one to locate it, and a third to make the necessary analysis, possibly all three have to call in number four whose function it is to quickly apply the remedy, whether medical or surgical.

The dream of a plant of this kind running automatically and smoothly, taking in a given weight of ore daily, turning out the appropriate amount of metal ready for shipment, is beautiful; but alas, it *is* a dream. That is, unless you have a very large force of human experts with both brains and hands to assist the "automatic" running.

Consider the complexity of the case (independent of the particular process) which must always include so many different functions that their mere enumeration is tedious. And of all these functions there is not one that does not demand chemical knowledge, quick perception and skillful manipulation.

You must know that your original solution is practically complete. You must know the same of each precipitation. You must ascertain the purity, as you go, of each product, whether final or intermediate. You must carefully avoid any excess in the use of chemicals.

On the mechanical side, exercise an eternal vigilance for leaks whether liquid or gaseous. Keep your batteries always at efficiency of 100, if you know what that is. Don't let anything get cold, or you will check the sequence of everything. But keep your fuel down to just the necessary point so as to get the right amounts of heat in the right places, with nothing going off by radiation or through the chimneys. (All this is SO EASY!)

Then, as a final little addition to all your duties, modify your daily procedure according to the variations in your raw material. That last is a mere trifle, of course. It doesn't add more than 40 of 50 per cent to the "regular" schedule.

The "Endless Chain" of Reagents

We shall end our survey with the general statement that nearly every process calculates on the regaining of reagents and very few ever realize that ambition. There are chemical, physical and financial reasons why this is impracticable. In the hyposulphite process, largely practiced in Mexico, and far from unknown in the United States, which is one of the most successful of all wet processes, neither the hypo nor the sodium

sulphide is regained entirely, although the chemistry of the process would indicate the full restoration of the former. The point is always reached at some stage, when a careful estimate will show that a given solution cannot be "regenerated" at a profit. In the case of salt, it is too cheap a chemical to waste money on, and it rarely pays to evaporate a salt solution merely to regain the solid.

Finally, we have not written this sketch to discourage the attempts at metallic extraction by wet methods. Our object has been rather to indicate in a very broad way, that some of the prevailing fallacies have been considered rather in an economic than a scientific sense.

There have been far more failures than successes. Throwing out all of the mere "tricks" and all of the processes which, although honest were based upon false chemistry, i. e., fundamentally false, there remains a number of failures, owing their downfall chiefly to matters of detail, which a more careful study of conditions might have foreseen. We believe that we have indicated the nature of most of these, and we are quite sure that many processes have failed in practice for lack of figuring. It is one thing to put through a chemical equation or a series of them, in the laboratory. It is quite another to carry out these transformations on the commercial scale, and bring time, space, apparatus, supervision and product into "one harmonious whole."

The Advancement of the Mercury Arc Rectifier

By Harry F. Perkins.

The extensive use of the mercury-arc type of rectifier has caused numerous articles to be published in this country and abroad, describing the characteristics and application of this form of rectifier. With the arc contained in a glass tube, the rectifier has been designed to operate series arc lamps at voltages up to 7500 direct current, with current over 6 amp, and for such other low-voltage direct-current purposes as charging storage batteries and operating small direct-current motors, where the current does not exceed 50 amp.

Within the last few years much work has been done toward increasing the capacity of the rectifier, and articles giving in part the results of this work have appeared from time to time in the *Elektrotechnische Zeitschrift*, Nov. 7, 1912; the London *Electrical Review*, Dec. 27, 1912; *Bulletin* de la Société Internationale des Elec., January, 1913; *A. E. G. Zeitung*, June, 1913; *Electrical World*, and other journals.

The object of this treatise is to show a recent form of iron tube which is used in place of glass in the development of the rectifier for high power, and which has been reduced to an extremely simple and stable state.

In Fig. 1 a rectifier capable of delivering 150 amp at 350 volts direct current is shown, combined with its vacuum pump on a common base. The cut shows practically all there is to the apparatus. The anodes, which are of steel, are insulated by heavy porcelain from the arms of the tube, and the pressure on their gasket surfaces, which prevents leakage and allows of expansion, is kept fairly constant by means of springs on their retaining bolts. A window is provided in a convenient place so that the interior may be viewed during operation.

It has not seemed advisable so far to do without the vacuum pump, for though the rectifier shows no great amount of leakage over long periods of time, the pump is conveniently available in case of any repairs, while it does not need to run during operation of rectifier. Only one pump would be required to a station, as it is only used occasionally and could handle several tubes at a time in an emergency. This pump requires about 1/3 hp to operate.

The rectifier carries the load previously stated without artificial cooling of any kind; it is started in practically the same manner as all low-potential tubes, as will be described later.

The particular tube shown in cut has been in operation for several weeks in a public garage, charging the batteries of a number of electric vehicles at a time. The charging takes place mostly at night, though occasionally some charging is done at noon, and over-charging is done once a week. A tube similar to this one has been in operation for the past three months. The weight of the tube is about 350 lb.

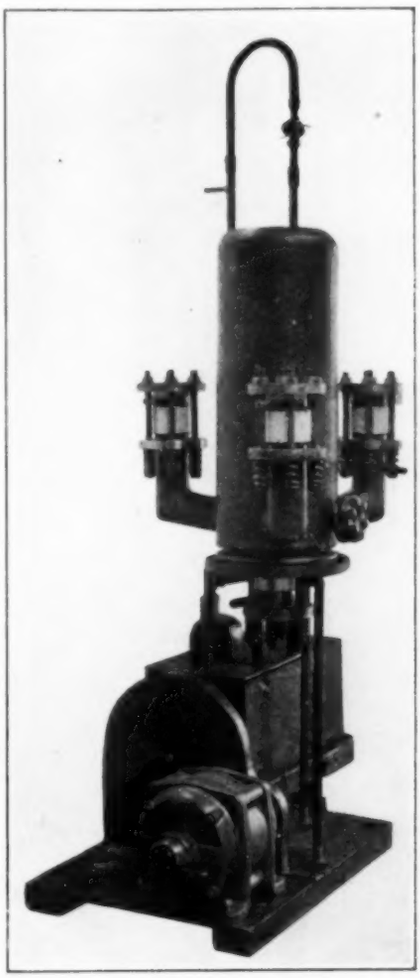


FIG. 1.—BATTERY CHARGING RECTIFIER FOR 150 AMPERES

In Fig. 2 a night view of the garage gives an idea of the rectifier load, the light spot on the tube shown in the background indicating that the tube is working. Some of the vehicles in the picture have lead batteries and some are equipped with Edison batteries.

The wiring connections are shown diagrammatically in Fig. 3, the rectifier being of the three-phase, three-arm type. A bank of three delta connected standard transformers step the line potential of 2200 volts, 60 cycles, down to 236 volts, which is applied to compensators connected in Y to give a neutral point, which is the negative side of the direct-current circuit.

The compensators also allow of voltage variation by means of taps. Close voltage regulation may be obtained by varying the tap connection one phase at a time. The slight unbalancing of phases thus produced does not affect the rectifier in the least. Each battery has a standard charging rheostat in series which gives the desired individual control. This gives to the rectifier all the regulation qualities of a direct-

FIG. 2.—RECTIFIER CHARGING BATTERIES ON SIX VEHICLES

current generator, without the rheostat losses. Very fine regulation can be obtained by using a standard induction regulator.

The arc rectifier is an apparatus having characteristics which are entirely different from either a generator, with its moving parts, or an alternating-current static transformer. There is no friction, windage, nor appreciable iron loss, nor I^2R losses of the usual kind in the rectifier; the loss is represented by a practically constant voltage drop, most of which is at the anode and cathode.

While it has often been stated that the voltage drop is constant, it is not found to be accurately so, but on the contrary, it is seldom steady. The variation is negligible, however, except in cases of very low voltage or small currents.

It is found that an increase of pressure in a tube will cause an increase in the voltage drop, and the general design of the

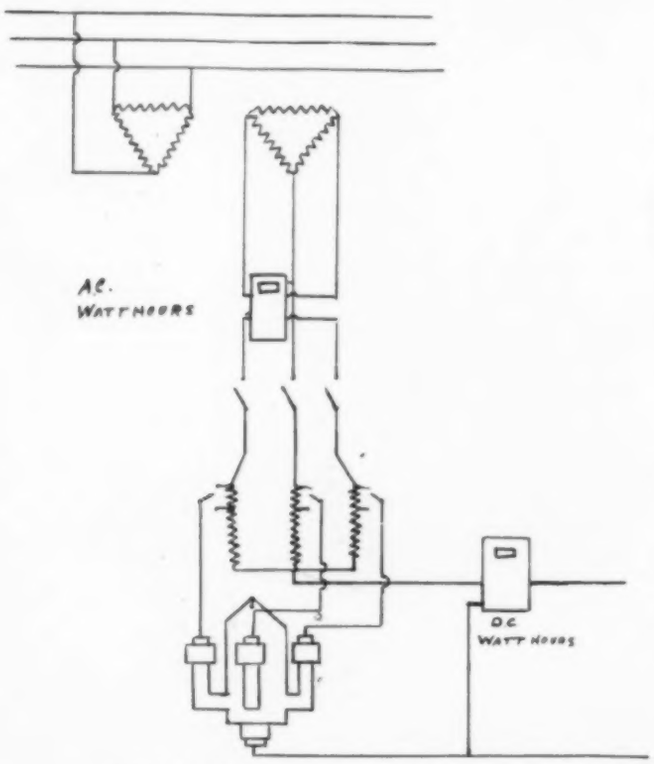


FIG. 3.—CONNECTIONS FOR RECTIFIER IN FIG. 2

tube has a considerable effect on it. The arc drop reaches 30 volts in some cases, while in moderate low-voltage, small-current apparatus, it is approximately 15 volts. Therefore the efficiency of a tube on 110 volts direct current ought to be about 85 per cent, with a decided improvement at higher voltages. That is actually the case.

A comparison between the efficiency of a motor-generator set and a low-voltage rectifier is shown in the curves of Fig. 4 (the slight gradual drop in rectifier curve is due mainly to drop in compensator). The advantages of the rectifier over the motor-generator are at once apparent, as the rectifier maintains its efficiency at all loads, and it is free from loss when no current is being taken. With rectifiers for higher voltages an efficiency of 97 per cent can be obtained.

The power factor created by the rectifier on the power circuit depends largely on the transformer design. The three-arm type of tube has a tendency to produce a unidirectional magnetic flux, which in turn distorts the current wave, and thus the power factor of the circuit is affected. By the proper arrangement of transformer a good power factor can be obtained and in the case of both single and polyphase rectifiers a power factor well above 90 per cent has been obtained.

This leads us to say something about the wave shape of current and voltage connected with rectifiers. Figs. 5 and 6 illustrate the pulsations delivered by the rectifier in Fig. 1. If the supply frequency is multiplied by the number of arms,

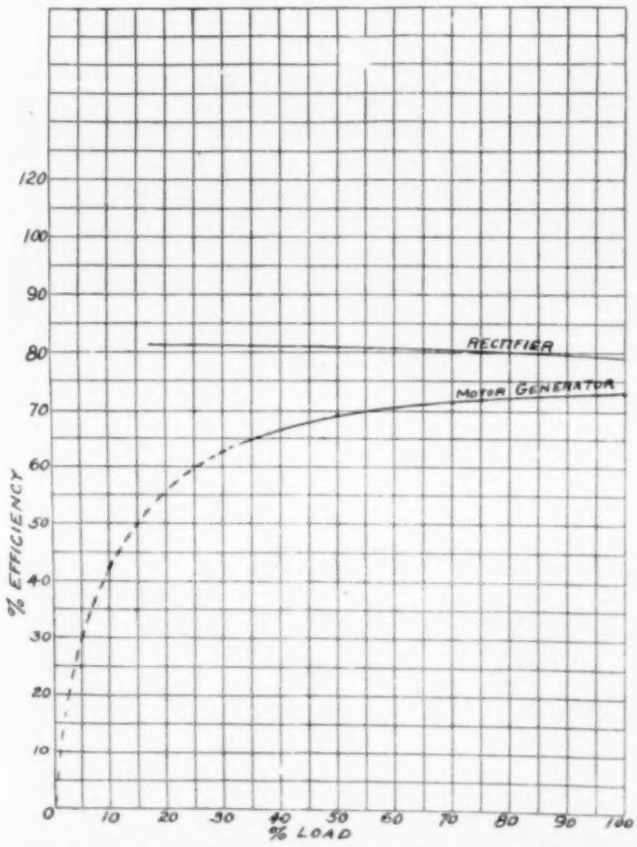


FIG. 4.—EFFICIENCY CURVE OF RECTIFIER IN FIG. 1 COMPARED WITH THAT OF MOTOR GENERATOR SET

the pulsations per second will be the result. Thus if six arms were supplied from a three-phase source, the hollows between the waves here shown would be filled each by a similar wave, and a current practically equivalent to that from a commutator can be obtained.

Figs. 7[a], 8 and 9 are fair samples of the rectified current from single phase with and without direct-current reactance, and of quarter-phase and six-phase rectifiers without addi-

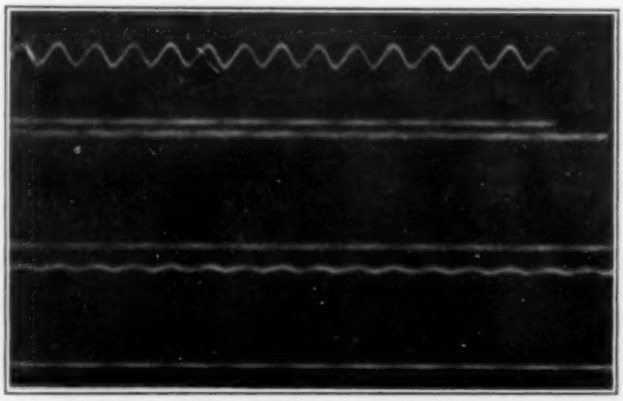


FIG. 5.—UPPER CURVE RECTIFIER VOLTS. MIDDLE CURVE BATTERY VOLTS. LOWER CURVE D. C. AMPERES

tional reactance. Fig. 7[b] shows the wave shape when running in multiple with a rotary converter.

While the starting and operating of the mercury arc rectifier is fairly well known, on account of its extensive use in small units, a detailed explanation may not be out of place here.

Taking the case of a single-phase tube (Fig. 10), it is noted

that there are two main anodes, a cathode and a starting anode. The main anodes are connected one to each end of the transformer winding, the cathode is connected to the positive (+)

FIG. 6.—UPPER CURVE BATTERY VOLTS, LOWER CURVE LINE VOLTS ON ONE PHASE

side of the direct-current circuit to be supplied. The negative (—) side of the direct-current circuit is connected to the middle point of transformer. The starting anode is connected through resistance to one of the main anodes.

When it is desired to start the tube, the starting anode is

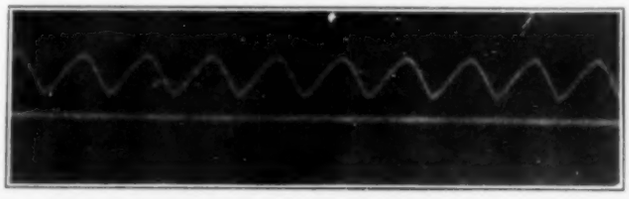


FIG. 7A.—SINGLE PHASE WITH REACTANCE IN CATHODE LINE

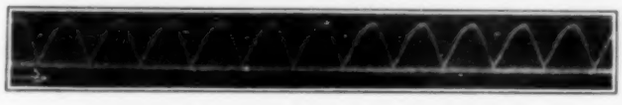


FIG. 7A.—SINGLE PHASE WITHOUT REACTANCE

brought into contact with the cathode. This allows an alternating current to flow, limited by the resistance mentioned above and the auxiliary load. When the contact is broken an arc will start and its direction will be determined by the potential direction at the instant of break. If, however, the starting anode happens to become cathode at the instant of break, the arc dies out when the potential reaches zero, and therefore lasts only a portion of half a cycle.

FIG. 7B.—SINGLE PHASE IN PARALLEL WITH ROTARY CONVERTER, CURVE OF D. C. VOLTS

Should the contact break when the potential is in the proper direction an arc is started with a cathode spot in the pool of mercury forming the cathode. This cathode spot, during its brief existence of less than half a cycle, is usually sufficient to ionize enough mercury vapor to start an arc from the main

FIG. 8.—QUARTER PHASE, UPPER CURVE, ANODE AMPERE ONE ANODE LOWER CURVE, D. C. AMPERES

anode opposite to the one connected to the starting anode, when the potential is reversed.

It is true that the arc will not start without ionization and

FIG. 9.—SIX PHASE VOLTAGE CURVE

that it will go out the instant that this ionization ceases. Therefore, as the arc will supply its own ionization while the voltage remains in the proper direction, it is important to have some means of continuing this ionization while the potential is falling below that necessary to maintain the arc, through

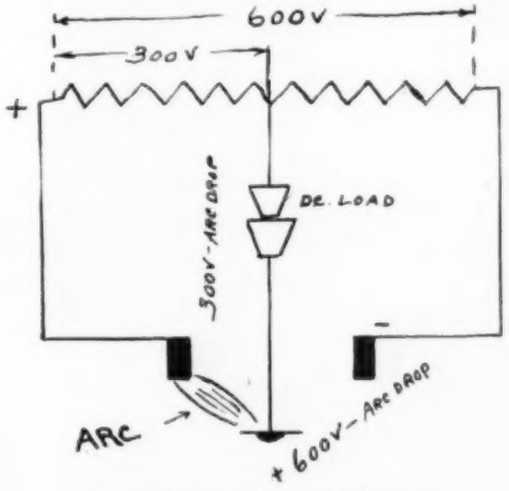


FIG. 10.—DIAGRAM OF CONNECTIONS

zero, to a potential that will start the arc from the other anode.

In the case of rectifiers supplying variable loads where the current may be off for short periods, as, for instance, a railway load, this carrying over of the arc is accomplished by

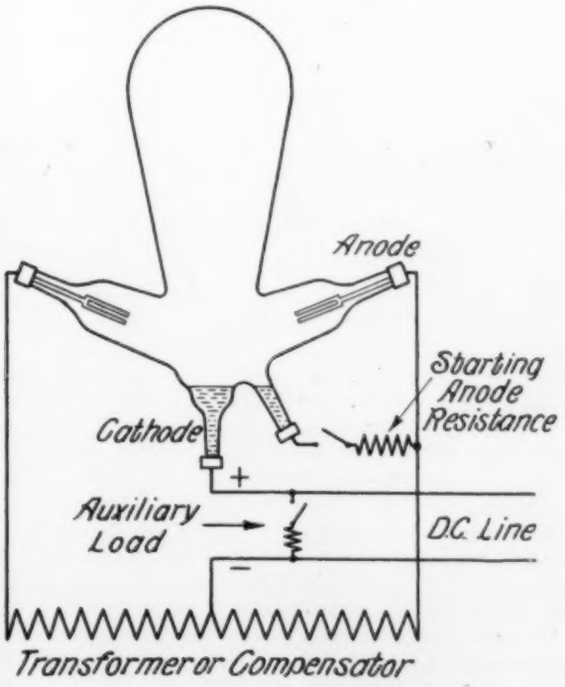


FIG. 11—DIAGRAM OF CONNECTIONS

supplying the starting anode from a separate direct-current source, or by inserting two small auxiliary anodes which can form part of a separate rectifier circuit supplied by an addi-

tional coil from the main transformer, or by a separate small transformer. This auxiliary rectifier can be designed to consume a minimum current at low voltage and can thus be made self-sustaining with small loss.

In rectifiers carrying a steady load this carrying over can be done to best advantage by placing in the direct-current circuit an air-gap reactance.

The main arc flows in one direction only when the cathode is excited and when the potential at the anode is positive with respect to the cathode, but no current will flow without excitation until a potential is reached that will cause a Geissler discharge, for the vacuum in these tubes is of a very high degree and ordinarily it would take several thousand volts to cause a discharge without a cathode spot. This is one reason why the current does not flow in both directions.

Upon examination of the diagram, Fig. 11, it will be seen that current flows in only one-half of the transformer winding at a time, and the potential of the direct-current load is that of one-half of the average (not the r.m.s.) transformer voltage, minus the arc drop; as the cathode is only separated from the working anode by the arc drop, we have practically all the potential of the transformer applied between the cathode and the idle anode, which is now negative with respect to the cathode.

That the rectification is perfect can be understood when it is shown that many hundreds of tubes are in service with over 10,000 volts between anodes.

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Quantitative Spectrum Analysis

By G. A. Shook

I. Colorimeters

There is to-day unfortunately no satisfactory explanation of the ultimate nature of light. In all probability, however, light is some sort of a transverse wave motion, but the character of the medium through which the waves are propagated and the mechanism by which the waves are produced are questions still unanswered. Nevertheless the experimentally derived laws in this field are so well established that the lack of a fundamental theory is not felt practically.

When white light passes through blue glass it becomes blue. Whether the white light contains, as one of its components, the blue color or whether the blue glass manufactures the blue light out of the white, we are not able to decide with so much confidence as the ancients or the philosophers of Newton's time. Yet the fact remains and there is never any exception to it.

We are still farther from satisfactory theories when we come to the case of colored solutions, due to the numerous exceptions to all the simple hypotheses that have been proposed. These hypotheses are probably correct, but due to such disturbing influences as incomplete ionization and chemical reaction they cannot be universally verified. Nevertheless as far as practical colorimetry is concerned, experimental errors are generally greater than errors that might arise due to the departure from fundamental colorimetric laws.

When a colored compound is dissolved in a colorless solvent the extent to which the solution is colored depends upon the concentration, or the mass of the solute per unit volume of the solvent, and there is a very simple relation between the two factors. A thickness of 10 cm. of a 20 per cent solution will produce the same light absorption as 20 cm. of a 10 per cent solution of the same compound, etc.

If the molecules of the dissolved substance are not dissociated and if furthermore the solvent exerts no influence upon the color, then this simple relation holds rigorously. However, if the dissolved compound is ionized or dissociated then there are several possible results.

1. If either the positive ion or the negative ion alone is colored and if it has the same color as the molecule then the law still holds even with partial ionization. The negative ions Cl, NO_3, SO_4, etc., are colorless and the copper salts formed with these ions are all blue. The positive ions K, Na, Br, etc., are colorless and the chromates formed with these metals are all yellow.

2. If the molecule and the colored ion do not have the same color then the rule would not hold. A very concentrated solution of copper chloride is green, but on dilution the color changes to blue and remains blue as long as the dilution continues. Consequently the law would hold if we considered only dilute solutions.

3. In the case of two colored ions, neither ion being colored the same as the molecule, for example H_2, Fe, Cu_2, the law would not hold except for dilute solutions, i.e., complete ionization.

The color of the ion may be different for different electrical charges; for example, the iron ion in the ferrous condition is green, while in the ferric condition it is yellow.

It might seem at first sight that any method for determining the concentration of a salt by means of its color would be unreliable, but in practice all these difficulties are not met with at once. In most cases we are dealing with dilute solutions or at least with a limited range of concentrations. It will be subsequently shown that in refined work it is not difficult to determine just how far the law holds for any particular case.

Law of Light Absorption

Whenever light falls on any absorbing medium, for instance, a piece of colored glass or a cell containing a colored solution, part is reflected, part absorbed and the remainder transmitted.

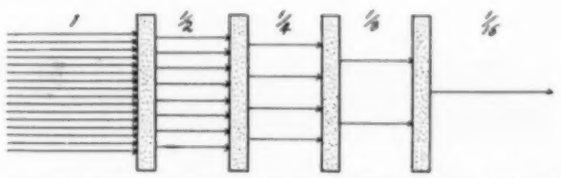


FIG. 1.—GRAPHICAL REPRESENTATION OF THE LAW OF ABSORPTION OF LIGHT

Let the intensity of the light incident upon a layer of unit thickness of an absorbing medium be I and the intensity of the transmitted light be I_1. If the coefficient of transmission is a, we may write

$$\frac{I_1}{I} = a \text{ or } I_1 = I\,a$$

a is that fraction of the incident light which is transmitted by unit thickness.

Now if this transmitted light traverses a second layer of unit thickness it will again be diminished in the same ratio, hence

$$I_2 = (I_1\,a)\,a = I\,a^2.$$

For three layers

$$I_3 = (I\,a^2)\,a = I\,a^3.$$

Therefore, if the light traverses a thickness of h units the transmitted light, I_h, has the value

$$I_h = I\,a^h \qquad (1)$$

For example, if the coefficient of transmission a is ½, then the intensity of the light transmitted through one layer of unit thickness is ½ the intensity of the incident light. If this transmitted light traverses a second layer, ½ is again transmitted so that we have left only ¼ of the original amount of light. If the light passes through three layers we have ⅛ left and for four layers only 1/16 or $(½)^4$ remains, etc. (Fig. 1.)

For a second absorbing medium

$$I_{h'} = I\,a'^{h'}$$

and if the transmitted light is the same in each case $I_{h'} = I_h$, whence

$$a^h = a'^{h'}. \qquad (2)$$

Now let b represent the transmission coefficient for a unit layer of a colored solution of unit concentration (1 g. of

solute dissolved in 1 cc. of solvent). We may now write

$$I_1 = I\ b,$$

and if the concentration is doubled, i.e., if the number of absorbing molecules is doubled, then

$$I_2 = (I\ b)\ b = I\ b^2.$$

Therefore, if the light passes through a unit layer of concentration c it follows that

$$I_c = I\ b^c \text{ or } \frac{I_c}{I} = b^c \qquad (3)$$

For a different concentration c' of the same dissolved substance we may write

$$I_{c'} = I\ b^{c'}$$

Since we have unit thickness in each case, $h = h'$, whence

$$I_c = I\ b^c = I\ a$$

and

$$I_{c'} = I\ b^{c'} = I\ a'$$
$$b^c = a \text{ and } b^{c'} = a' \qquad (4)$$

From (1) it follows that

$$\frac{I_h}{I} = a^h = (b^c)^h = b^{ch}$$

This is known as Beer's Law.

Combining (2) and (4) we obtain the relation

$$b^{ch} = b^{c'h'},$$

whence

$$c\ h = c'\ h' \qquad (5)$$

Equation (5) forms the basis of the theory of all colorimeters. Light of the same intensity is caused to pass through two suitable vessels; one contains a solution of known concentration c' and the other a solution of unknown concentration c of the same solute. The heights h and h' of the two columns are varied until the transmitted light through each is of the same intensity and then the heights are measured. The unknown concentration may then be determined by means of the simple relation

$$c = \frac{c'\ h'}{h}$$

There is another class of instruments which is used for determining concentrations, known as spectrophotometers. A spectrophotometer simply compares the intensity of two light sources for some particular color. From (3) it is seen that if the ratio of the transmitted light to the incident light is known, the concentration may be estimated. Equation (3) may be put in the form

$$\log \frac{I_c}{I} = c \log b = k\ c \qquad (6)$$

where k is a constant for any particular wavelength of light and any particular substance in solution.

Equation (6) simply means that the logarithm of that fraction of light transmitted by a unit layer of a colored solution is directly proportional to the concentration of the solution. This unit thickness may be 1 cm. or any other convenient width of containing cell.

For simplicity we will call the negative logarithm of the ratio $\frac{I_c}{I}$ the *extinction coefficient*, so that

$$e = -\log \frac{I_c}{I}.$$

The logarithm of a fraction is, of course, negative, hence, by prefixing the negative sign, e becomes a positive quantity. Equation (6) may now be written

$$c = A\ e$$

We will moreover call the constant A the *absorption ratio*.

The form of the expression for the determination of e depends, of course, upon the type of instrument used, and this will be fully considered in a succeeding issue of this journal.

The extinction coefficient is sometimes defined as the reciprocal of the value of the thickness of a layer such that the incident light is diminished to one-tenth its value. This, however, is an unnecessary complication since all that is required is that we may be able to express the ratio of the intensity of the transmitted light to the intensity of the incident light.

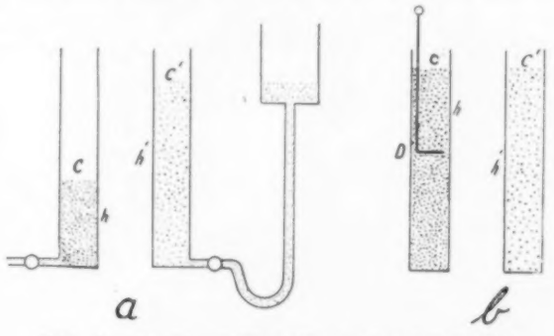
There are five essential parts to a good colorimeter, viz.:

1. Containing vessels for the standard and unknown solutions.
2. A method for adjusting the height of one or both of the two liquid columns.
3. An accurate means for measuring the heights of the two columns.
4. A device for bringing the two fields of view close together in order to determine when equality of brightness is obtained. and
5. A source of illumination.

The simplest type of colorimeter, but one which is too crude for accurate work, consists merely of two cylindrical glass jars or graduates placed upon a white mat surface, to give uniform illumination. One graduate is filled to a convenient mark with the unknown solution and a standard solution is poured into the other graduate until the two top surfaces of the liquids appear equally bright. Knowing the two heights the unknown concentration may be readily calculated.

Of course, the standard solution may be filled to a given mark in one cylinder and the unknown poured into the other cylinder until a balance is effected.

Sometimes one jar is filled to a given mark (100 cc.) with an unknown solution and the other jar is filled with a stronger solution of known concentration to a given mark (10 cc.). The known solution is then diluted until both jars appear to have the same intensity of color and the required amount of



FIGS. 2A AND 2B.—SIMPLE TYPES OF COLORIMETERS

dilution necessary to effect this balance can be estimated by observing the volume of the standard solution.

In such cases the diameter of the two cylinders must be equal, but when we are concerned only with the heights of the liquid columns this is not necessary.

A slight improvement upon this simple type of colorimeter is to provide the jars with stop cocks for drawing off the solutions. If one of the jars, say the one containing the known solution, is connected to a reservoir, Fig. 20, which can be gradually raised and lowered, it will greatly facilitate the operation of making an intensity balance.

A still better arrangement is a tubular reservoir provided with a glass plunger, in which case the solutions come in contact with glass only.

Instead of changing the height of the liquid a white disc D, Fig. 2*b*, may be lowered into one jar, thus changing the effective length of the liquid column.

Another scheme which is often used is to fill a number of jars, all to the same mark, with standard solutions of varying concentrations. A similar jar is then filled to the same mark with the unknown solution and its concentration may be roughly estimated by comparing the intensity of its color with the standard solutions.

The chief objection to all of these methods is that only a rough estimation of the equality of brightness can be made due to the fact that the two fields of view are not close enough together. This objection is overcome in the following instruments:

Duboscq Colorimeter

This instrument is shown in schematic form in Fig. 3. The standard and the unknown solutions are placed in two glass

jars, A and A', into which dip two solid glass cylinders, B and B'. Each cylinder is actuated independently by means of a rack and pinion, and to each pinion is attached a fiducial mark I which moves over a graduated scale. The heights h and h' may be read directly from the scales.

Daylight is diffusely reflected from an opalescent mirror M to the jars A and A', is then transmitted through the cylinders B and B' and by means of the reflecting prisms P and P' finally reaches the eye.

By means of these two prisms, which are cemented together, the two fields of view are brought in close contact so that the observer can make an accurate intensity balance. The faces of the prisms, the ends of the cylinders and the bottoms of the jars are all optically plane, so that the field of view is very clear and uniform, which is essential, if accurate work is required.

The eye is not able to make any estimate whatever of the relative brightness of two illuminated fields, but it can accurately determine when two fields are equally bright if favorable conditions obtain.

The two fields must be uniformly illuminated throughout and must not be separated by a line of any appreciable width. When the eye is required to move from one field to the other, as in the case of the crude colorimeters, it loses the impression of one field before reaching the other and the accuracy of the judgment is greatly impaired.

In some forms of colorimeters both eyes are used, but such a method is unreliable.

It is moreover desirable to have just sufficient light enter the eye to prevent fatigue. If the field is too dark the eye is constantly under a strain due to an abnormal dilatation of the iris and if it is too bright an abnormal contraction is necessary.

To make an accurate intensity balance, means must be provided for varying the intensity of one or both of the fields in a continuous manner. One field should be made alternately brighter and darker than the other, diminishing each time the difference in intensity until no difference can be detected. Care must always be taken in making a setting, but the operation of making one field brighter than the other must not be prolonged until the field appears blurred, for then the eye has lost its maximum sensitiveness.

A colorimeter similar to the Duboscq, due to Stammer, is sometimes used for beer analysis. The standard solution is here replaced by a colored glass and the containing vessel is moved instead of the dipping cylinder. This instrument, however, is not applicable for all adulterations.

Müller has designed a similar apparatus for water analysis. It has a 10 cm. tube which is filled with water, to which Nessler's reagent has been added. The balance is made by means of gold-colored glasses 1.5 mm. in thickness. The equivalent effect of the glasses is given in the following table:

1 glass is equivalent to	.02	mg. of NH_3 in 100 cc. of H_2O.
2 glasses are equivalent to	.05	
3 glasses are equivalent to	.07	
4 glasses are equivalent to	.10	
5 glasses are equivalent to	.15	
6 glasses are equivalent to	.20	

Wolff Colorimeter

The essential difference between the Wolff colorimeter and the Duboscq is that the former is not provided with the dipping cylinders. In order to bring the two fields to the same intensity, liquid is drawn off from either vessel.

It is somewhat easier to calculate the concentration if one jar is filled with the weaker solution up to the 100 mark and the stronger solution drawn off. This arrangement has the disadvantage that one cannot make a fine adjustment of the two fields as in the Duboscq. In order to repeat a setting it is necessary to pour some more solution back into the vessel or to disturb the solution which is at the 100 mark.

Since the accuracy of the concentration, finally determined, depends primarily upon the accuracy of making a good photometric balance, this is a weak point in this type of apparatus. It does, however, have this slight advantage, that the jars do not have to be made as long as the ones used in the Duboscq instrument since there is no liquid to overflow.

In some types of the Duboscq colorimeter, this difficulty is overcome by making the upper part of the jars of much larger diameter than the lower part.

In the Duboscq instrument, the light entering the two vessels does not come from the same part of the mirror and, therefore, it is necessary to bring the two fields to the same intensity before the solutions are poured into the vessels. In the Wolff instrument, however, the light which reaches the two vessels is reflected, by means of a Fresnel double prism, from approximately the same region so that the two fields are practically always the same in intensity.

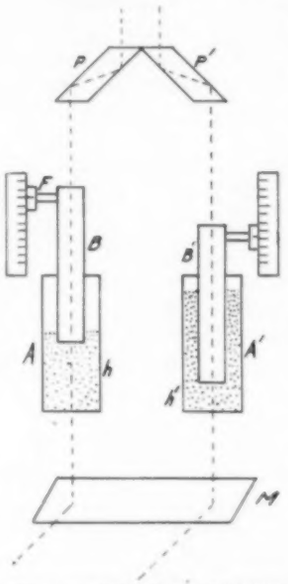


FIG. 3.—DUBOSCQ COLORIMETER

A smoked glass is sometimes used to diminish the intensity of the field, thus enabling the observer to make a better balance. The bottom caps of the two glass tubes may also be removed for cleaning.

The above types of colorimiters involve the essential features, although there are a number of other colorimeters which employ different methods for changing the intensity and different devices for producing good fields.

The most sensitive photometric screen is undoubtedly the Lummer-Brodhun. The optical system for bringing the two fields together, when this photometric screen is employed, is shown in Fig. 4.

L is the Lummer-Brodhun prism and C is a compensating cube, which is inserted in order that the path of each beam may have the same optical length. This arrangement could of course be used on any colorimeter employing two vertical vessels.

Extinction Colorimeters

There is another class of colorimeters sometimes employed for turbid solutions. Instead of comparing the intensity of the light transmitted through a standard solution with the light transmitted through an unknown solution the light through a single column is made extinct by varying the length of the column.

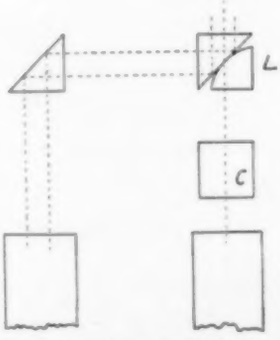


FIG. 4.—LUMMER-BRODHUN COLORIMETER

A turbid solution of known concentration is placed in a suitable vessel and the observer views a standard candle or a lamp of constant intensity through this column of absorbing material. The length of the column is increased by pouring in more of the turbid solution until the light just becomes extinct and then the length of the column is noted. If the same operation is carried out with an unknown solution its concentration may be easily calculated.

Writing Beer's Law for the two cases we obtain

$$\log \frac{I_h}{I} = b^{ch}$$

and

$$\log \frac{I_{h'}}{I} = b^{c'h'}$$

But the fraction of the incident light which reaches the eye in each case is approximately the same, hence

$$\log\frac{I_h}{I} = \log\frac{I_{h'}}{I}$$

$$c\,h = c'\,h' \qquad (7)$$

Absolute darkness is not, of course, produced in each case when the light disappears, consequently I_h is a finite quantity, though very small.

This method has the advantage that no standard solution is required and the necessary apparatus is much simpler than that which is necessary for the comparison of the intensities of two fields.

Colorimeters based upon this principle cannot, however, be used for accurate work since one is not able to reproduce the extinction point with the same accuracy that one can reproduce the equal-intensity point.

Moreover, the personal equation must be reckoned with, since different eyes vary greatly in this respect.

Again, the standard lamp must be kept constant, which is not always possible without complicated auxiliary apparatus.

This method has been used extensively for the rough estimation of the amount of sulphate in water, urine, coal, cement, etc., and in fact in any case where a fine precipitate could be easily formed.

The early investigators using this simple method observed that the product of the concentration and the height of column necessary to produce extinction is a constant, but aside from this they gave no theory of the method. Some of them state that it is necessary to keep the light constant and others state that this precaution is unnecessary. The column of liquid must always be lengthened until the lamp just disappears, but manifestly a strong light will require a greater length of absorbing material than a weak light.

Evidently it is not necessary to restrict this class of analysis to the extinction colorimeter since any of the determinations may be made by means of a spectrophotometer with much greater acuracy. The colorimeter is not so well adapted to this sort of work as the spectrophotometer, due to the fact that the former requires a much longer layer of absorbing medium and the consequent loss of light is too great.

There are a number of colorimeters on the market which are designed for special purposes, but the fundamental principle involved in most of them is the same and consequently one instrument, with slight modifications, can be used for most all purposes.

There is no one colorimeter which embodies all of the best features. The best method for bringing the two fields together is that shown in Fig. 4, while the best method for varying the intensity and determining the heights is that used on the Duboscq colorimeter.

The containing vessels should be easily removable for refilling and cleaning.

Since we are concerned only with the thickness of the absorbing layer and not with the volume of the solution, it is not necessary that the two vessels have the same diameter nor that either has a uniform cross-section.

Metallic scales and delicate optical parts should be well protected from the action of salts.

Before taking any observations upon a solution one should make sure that both fields of view are equally illuminated. They can be brought to equality by shifting the colorimeter slightly or by rotating the illuminating mirror.

In the next article of this series spectrophotometers will be considered.

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Plumbago is the chief mineral product of Ceylon, and the United States is the principal consumer of the exported mineral. The mined product is sorted by hand and classified into "lump," "chip," "dust," and "flying dust." These sorts are then graded according to quality.

A Metallurgical Problem

Concentration and Reverberatory Smelting of a Second-Class Copper Ore

By F. W. Traphagen

One of the most interesting of all metallurgical calculations is that involved in the treatment, by reverberatory smelting, of a second-class copper ore.

With proper attention to details and a careful consideration of what occurs in good practice, one can calculate with great precision what will take place in the handling of such an ore, from mine to marketing of finished products; and can make also a very close estimate of costs and losses, hence of the commercial promise of each particular deposit.

These estimates involve a careful study of local conditions and a close comparison with practice in various places, but when made with care one can justly have great confidence in the results of his calculations. Of course a critical analysis of local conditions and a study of the results of experimental work, especially of concentration, are very important considerations.

In the particular problem taken for illustration of the methods for preliminary calculations, an attempt is made to secure fairly close approximations to average conditions. The data used will fit Montana practice more nearly than that of any other section, but an attempt is made to avoid, as far as practicable, the localization of data.

Just where the line of demarcation between first and second-class ore shall be placed depends upon three principal factors: the market price of copper; the respective losses in concentration with roasting and reverberatory smelting, as contrasted with that in direct blast-furnace smelting; and the costs of the respective treatments.

With a high market price and with large concentration losses, the percentage of copper marking the line of demarcation is lower.

Assumed Metallurgical Conditions

In our calculations we will assume the following:

Second-class ore for concentration, roasting, reverberatory smelting and converting treatment will contain: Copper, 4 per cent; iron, 11 per cent; sulphur, 14 per cent; silica, 58 per cent; silver, 2 ounces per ton; gold, 0.015 ounce per ton, and alumina and other non-metallic gangue material, balance, approximately 13 per cent.

Reverberatory slag: Silica, 40 per cent; ferrous oxide, 45 per cent, and matte, 1 per cent, the precious metals being assumed to be present in the slag in the proportions in which they exist in the matte carried by it.

Converter slag: Silica, 29 per cent; ferrous oxide, 58 per cent; copper, 2 per cent; silver, 1.2 oz. per ton; gold, 0.005 ounces per ton. Inasmuch as it is the universal custom to retreat converter slag, either in blast furnaces or in reverberatory furnaces, with a high recovery; and also as this slag is a good basic flux for the blast furnace charge, we will in our calculation assume a final loss in converter slag of only 20 per cent of the values present, making no charge for recovery of metallic constituents.

Concentration losses will be assumed in this problem to be 15 per cent of all metallic constituents, including the associated sulphur. The alumina and undetermined gangue will be assumed to follow the silica in ratio to the amounts of each per cent in the original ore.

Second-class ore of the same composition as that given above will be used for "lining ore," *i.e.*, for use in the converter to supply the necessary silica for the oxide of iron formed from the matte during converting. No charge will be made for the treatment of this ore other than for its mining and freight to the smeltery.

It is assumed that a 40 per cent matte is required and that the blister copper contains 99.2 per cent copper.

In order to avoid unduly complicating the problem, volatilization and dust losses are not to be taken into account.

Assumed Costs and Prices

The following costs of treatment will be assumed:

Mining, including development, administration and general depreciation, per ton ore	$4.00
Freight from mine to smelter, per ton ore	.20
Concentration, per ton ore	.60
Roasting, per ton concentrate	.50
Smelting, per ton calcine	3.00
Converting, per ton matte	4.00
Freight, refining and marketing, per ton blister copper	26.00

The market value will be taken as 15 cents a pound for copper, $20.67 an ounce for gold and 60 cents an ounce for silver.

Required Data

The following information is required:

1. Composition and weight of tailing produced, *i.e.*, copper, per cent; gold and silver, ounces each per ton.

2. Composition and weight of concentrate, especially percentage of silica to be left with the concentrate, for the direction of the foreman in his work, and concentration ratio.

3. Composition and weight of calcine, and particularly percentage of sulphur to be left in calcine for production of matte of required copper tenor.

4. Composition and weight of reverberatory slag, and losses in pounds of copper and ounces of gold and silver.

5. Composition and weight of matte available for concentration, *i.e.*, weight of total matte from calcine minus that carried by reverberatory slag.

6. Weight of "lining ore" required to convert matte. This involves a calculation of the available silica, *i.e.*, the silica in excess of that required by the iron of the "lining ore" which goes into the converter slag in the same ratio as the iron from the matte. The weight of copper, gold and silver contributed by this ore to the total must also be determined and added.

7. Composition and weight of converter slag to determine losses of copper, gold and silver, 20 per cent of which is to be deducted.

8. Composition and weight of blister copper produced, and of the three metals, *i.e.*, per cent copper and ounces per ton gold and silver in blister, and total pounds copper and ounces gold and silver in this final product of the smeltery.

9. Costs of treatment and returns from the marketing of products.

10. Costs for production of a pound of copper, with and without elimination of gold and silver values.

One of the simplest methods of calculation is to base all weights and all costs on the original ton of ore.

The results of successive calculations will be tabulated, *assumed values being given in italics.*

Solution of the Problem

The first step consists in finding the weights of the metallic constituents of the ore left with the concentrate and tailing respectively, on the basis of the concentration loss. In this case 85 per cent will be in the concentrate, the remainder in the tailing. This is calculated in pounds.

In order to determine the amount of silica in concentrate and also in tailing, the quantity of iron available for fluxing purposes must be determined. This quantity varies with the grade of matte required. The higher the grade the less iron will be present in the matte; hence the more iron for fluxing purposes and the greater quantity of silica to be left in the concentrate.

In our example we have taken a 40 per cent matte. The copper is present as cuprous sulphide, Cu_2S, and the remainder is calculated as ferrous sulphide, FeS, neglecting the small amount of gold, silver and other elements which would make a very small percentage of the total. This assumption is not strictly true, for there is frequently an excess of iron over the assumed FeS; but it is sufficiently close for ordinary work.

In our calculations we will use, instead of the exact atomic weights, the nearest round numbers. Taking copper at 64 and sulphur at 32, Cu_2S would be 80 per cent copper and 20 per cent sulphur; and the 68 pounds copper recovered in our concentrate would require 17 pounds sulphur to form Cu_2S for total matte.

This 85 pounds of cuprous sulphide constitutes 50 per cent of a 40 per cent matte, *i.e.*, 40 per cent copper plus 10 per cent sulphur, to make Cu_2S, leaving a balance of 50 per cent for the ferrous sulphide, which in this case is equal to 85 pounds.

Ferrous sulphide, FeS, contains 56 parts of iron and 32 parts of sulphur in 88 parts of the compound, or in simpler terms 7/11 iron and 4/11 sulphur. This is equal to 54 pounds of iron and 31 pounds of sulphur.

Fifty-four pounds of iron being required for matte, the difference between this figure and the weight of iron in the concentrate gives us the amount available for fluxing silica:

$$187 - 54 = 133 \text{ lb. iron for flux.}$$

The reverberatory slag selected consists of 45 per cent ferrous oxide and 40 per cent silica. The 133 pounds of iron will form 171 lb. FeO, thus:

$$56\ (Fe) : 72\ (FeO) :: 133 : \times, \text{ or } 171 \text{ lb. FeO.}$$

The amount of silica required by this iron is

$$171\ (FeO) : \times\ (SiO_2) :: 45 : 40, \text{ or } 152 \text{ lb. } SiO_2.$$

This is the amount to be fluxed, and hence is the weight of silica in concentrate, calcine and reverberatory slag, the same weight being present in each.

The weight of alumina, etc., in concentrate would be:

$$\underset{SiO_2 \text{ in ore}}{1160} \quad \underset{SiO_2 \text{ in conc.}}{152} \quad \underset{Al_2O_3 \text{ in ore}}{26} \quad \underset{Al_2O_3 \text{ in conc.}}{x}$$

or 34 lb. alumina, etc., in concentrate, which is also the weight in the calcine and reverberatory slag.

Professor Peters[1] takes the view, justly I think, that iron in

[1] Principles of Copper Smelting, p. 31.

calcine is present as ferric oxide and ferrous sulphide, but not as ferrous oxide. Making this assumption, these two iron compounds react on each other during smelting thus:

$$3Fe_2O_3 + FeS = 7FeO + SO_2.$$

In modern reverberatory smelting the atmosphere of the furnace, so far as its influence on the charge is concerned, is neutral, for the charge spreads out at once on the surface of the matte left on the hearth and is covered or surrounded by the slag left from the previous skimming and is not affected by the gases above it.

There may be, of course, a small amount of magnetic oxide present, but the amount is generally so small that it may be safely ignored in our calculations. The ferrous oxide of the slag then is derived from the calcine in which it exists, not as ferrous oxide but 6/7 in the form of ferric oxide and 1/7 as ferrous sulphide. This, with our figures for other constituents, gives us the weight of calcine.

The 152 pounds of silica constitutes 40 per cent of our slag, the total weight of which is

$$40 : 100 :: 152 : x, \text{ or } 380 \text{ lb. slag.}$$

Assuming the matte lost in slag to be 1 per cent, we have in slag 3.8 pounds of matte which we distribute among its different constituents. Subtracting these results from the weights in total matte gives us weights going to the converter.

Our lining ore contains considerable iron which goes into slag in the ratio established, just as the iron of the matte does, and uses up a proportionate amount of silica. To determine the available silica of this lining ore we must first deduct the silica required by its own iron, which is 11 per cent, equivalent to 14.14 per cent ferrous oxide. According to assumptions made for the converter slag this will be:

$$\underset{\%FeO}{58} : \underset{\%SiO_2}{28} :: 14.14 : x, \text{ or } 7.07 \text{ \% silica.}$$

Deducting this amount from the 58 per cent silica in the ore, we have 50.93 per cent available silica.

We have 52.79 pounds of iron in our available matte which must be oxidized and fluxed. This is equivalent to 67.87 pounds ferrous oxide, and will require 33.935 pounds of silica,

TABULATION OF RESULTS
(ASSUMED VALUES IN ITALICS)

Material	W'ght, Lb.	COPPER		IRON		Ferrous Sulphide, Lb.	FERROUS OXIDE		Ferrid Oxide, Lb.	SILICA		SULPHUR		ALUMINA		SILVER		GOLD	
		%	Lb.	%	Lb.		%	Lb.		%	Lb.	%	Lb.	%	Lb.	Oz. per Ton	Oz.	Oz. per Ton	Oz.
Second class ore	2000	*4*	80	*11*	220	.....	...	....	.....	*58*	1160	*14*	280	*13*	260	*2*	2	*0.015*	0.015
Concentrate	679	..	68	...	187	.....	...	....	.....	22.4	152	...	238	...	34	5	1.7	0.037	0.01275
Tailing	1321	0.9	12	...	33	.....	...	....	.....	..	1008	...	42	...	226	0.595	0.3	0.00446	0.00225
Calcine	548.8	..	85	...	....	114.8	...	....	163	..	152	107	....	...	34	.....	1.7	.....	0 01275
Matte—total	170	*40*	68	...	54	.....	...	....	.....	..	.....	...	48	...	....	20	1.7	0.15	0.01275
Matte—in slag	3.8	..	1.52	...	1.21	.....	...	....	.....	..	.....	...	1.07	...	....	.....	0.038	.....	0.000285
Matte—available	166.2	..	66.48	...	52.79	.....	...	....	.....	..	.....	...	46.93	...	....	.....	1.662	.....	0.012465
Reverberatory slag	380	0.4	1.52	...	....	.....	*45*	171 .	.....	*40*	152	...	....	...	34	.....	0.038	.....	0.000285
Lining ore	66.6	*4*	2.66	*11*	7.33	.....	...	....	.....	58	38.63	*14*	9.32	*13*	8.66	*2*	0.0666	0.015	0.0005
Converter slag	133.2	*2*	2.66	...	....	.....	*58*	77.28	.....	*29*	38.63	..	....	..	....	*1.2*	0.08	*0.006*	0.000333
Blister copper	69.16	*99.2*	68.61	...	....	.....	...	....	.....	..	.....	...	....	...	....	49.54	1.713	0.37	0.0129

$$\text{or } \frac{100}{50.93} \times 33.935 = 66.6 \text{ pounds of ore.}$$

BALANCE SHEET

To Mining 1 ton ore @ $4..	$4.0000	By Cash from sale of metals:	
Freight 1 ton ore @ $0.20	.2000	68.61 lb. copper @ $0.15...	$10.2915
Mining 66.6 lb. ore (lining)	.1332	1.713 oz. silver @ $0.60....	1.0275
Freight 66.6 lb. ore (lining)	.0067	0.129 oz. gold @ $20.67....	.2666
Concentrating 1 ton ore @ $0.60	.6000		
Roasting 679 lb. concentrate @ $0.50 per ton...	.1679		
Smelting 548.8 lb. calcine @ $3 per ton	.9232		
Converting 166.2 lb. matte @ $4 per ton ...	.3324		
Marketing 69.16 lb. blister copper @ $26 per ton	.8991		
Total cost	$7.2625		
Profit	4.3231		
	$11.5856		$11.5856

Cost per lb. copper, $7.2625 ÷ 68.61 = 10.58 cents.
Cost per lb. copper, crediting value of precious metals ($7.2625 − $1.2941) ÷ 68.61 = 8.7 cents.

This ore will contribute gold, silver and copper, and these amounts must be added to the total..

Our 170 pounds of total matte contains 1.7 and 0.01275 ounces silver and gold respectively, hence the slag will contain:

170 : 3.8 :: 1.7 : x = 0.038 ounces silver, and
170 : 3.8 :: 0.01275 : y = 0.000285 ounces gold which must be deducted to give gold and silver going to converter.

From the 66.48 pounds copper in matte going to converter, plus 2.66 pounds copper furnished by "lining ore," is subtracted 20 per cent of the copper carried by the converter slag. The converter slag weighs 133.2 pounds, 2 per cent of which is 2.66 pounds; and 20 per cent of this last figure is 0.53 pounds, leaving 68.61 pounds of copper resulting from the treatment of one ton of ore, plus the required lining ore. In like manner the gold and silver are obtained, the results being given in the table.

One ton of ore produces 679 pounds of concentrate, hence the concentration ratio is 2000 : 679 :: 2.94 : 1.

The gold and silver are calculated to points beyond the possibility of assay, for the reason that these small fractions become of great consequence when multiplied by several thousand, and have a marked bearing on the sum total of profit.

Summary of Procedure

1. From percentage composition calculate pounds per ton of each constituent. Gold and silver, of course, in ounces per ton.

2. On basis of concentration loss, calculate weights of metals and of sulphur, in concentrate and tailing respectively.

3. From weight of copper in concentrate calculate weight of iron required to form matte.

4. Subtract iron required for matte from total iron in concentrate and calculate weight of silica required to flux this iron in ratio of ferrous oxide and silica in reverberatory slag. This gives weight of silica to be left in concentrate.

5. Calculate weight of alumina in concentrate on basis of ratio existing in ore.

6. The total weight of concentrate is now known. From this weight and the weight of silica present the percentage of silica can be determined.

7. The composition of the tailing can be determined by subtracting the weights of the constituents of the concentrate from those of the ore. The total weight of tailing is obtained in a similar manner, and from these data the percentage of copper and ounces per ton gold and silver can be calculated.

8. The weight of reverberatory slag is obtained from the weight and percentage of silica. From this weight we calculate the weights of copper, iron, gold and silver carried by the matte in the slag, and these weights subtracted from the corresponding weights in total matte give metals going to converter.

9. The weight of lining ore required is determined by first finding its available silica and dividing the weight of silica required to flux the iron in matte going to the converter by the percentage of available silica and multiplying the quotient so obtained by 100.

10. From the weight of lining ore used the weights of copper, gold and silver contributed to the total can be obtained.

11. The weight of converter slag can be calculated from either the iron or silica, the weights and percentages of which are known. From this weight and the determined recovery by retreatment of converter slag, losses of copper, gold and silver are calculated.

12. The above calculations give us the weights of the various products treated and the final recoveries of copper, gold and silver, from which the cost of treatment and value of products are obtained.

13. The weight of blister copper is determined by dividing the weight of copper by its percentage and multiplying by 100.

14. Ounces per ton gold and silver are obtained by dividing ounces present in blister copper by the weight of blister and multiplying quotient by 2000.

Denver, Col.

Successful copper smelting in blast furnaces at great altitudes requires that the capacity of blowers and air conduits, and the area of tuyeres, be increased with the lowering of atmospheric pressure. For an atmospheric pressure of 8.75 lb. per sq. in., these factors would have to be increased 70% in order to burn the same quantity of coke as in a furnace designed for sea-level conditions. Other features of smelting at high altitudes are (1) reducing radiation losses from crucible and jackets; (2) the use of hot blast, and (3) the use of back-pressure on the furnace in such a way that the fuel is burned under the mose desirable pressure for its economical working. These facts are deduced from a mathematical consideration of the subject by VINCENTE PAZOS Y SACIO, in the July *Quarterly* of the Columbia School of Mines.

The Elmore vacuum flotation process at the Sulitelma mines, Norway, produced 850 tons of copper concentrate in the month of July, 1913.

Cyanide Practice in the Black Hills, South Dakota.—III.

By H. C. Parmelee

Lundberg, Dorr & Wilson Mill

On July 31, 1913, the 100-ton cyanide mill of Lundberg, Dorr & Wilson, situated near Terry, S. D., definitely ceased operations after a long, profitable and interesting career. The plant was closed on account of the exhaustion of the ore supply in the firm's mines. Since January, 1904, this mill has been in continuous operation, with the exception of two periods of idleness during the "eight-hour" and "closed-shop" labor strikes in 1907 and 1910. During this period it has been used for the treatment of ore from the firm's Buxton and Bonanza mines, and has been operated also as a custom mill for other mines in the vicinity.

A general view of the property is given in Fig. 1. The Buxton mine is located on the north side of Fantail gulch, below the town of Terry, and the Bonanza is across the gulch on the south side. The mill was located on the Buxton side, and ore from that mine was trammed directly into the ore bins. The Bonanza ore was delivered to the top of the mill by a Bleichert aerial tramway, with buckets carrying 1200 lb. ore. There is only a single span of 910 ft. in this tramway, which has a capacity of 15 tons per hour and can be operated by one man.

History

The mines were opened in the early eighties, and as no recovery could be made by amalgamation or concentration, the first ore produced was shipped by wagon and railroad to the smelter at Omaha. The freight and treatment charges amounted to $40 per ton. In 1892 a smelter was built in Deadwood and the cost of treatment was reduced at first to $16 and gradually during the next 10 years to $8 per ton. After producing ore to the value of over $1,000,000, the Buxton Mining Co., owning

FIG. 1.—LUNDBERG, DORR AND WILSON MILL, TERRY, S. D. VIEW FROM MINE TERMINAL OF TRAMWAY

the property, stopped operations and turned the mines over to leasers. Later the property was sold for a nominal sum, as being entirely stripped of ore that could be profitably handled. The new owner leased it to John Lundberg, and for nearly two years the ore mined was treated at a custom cyanide mill operated by Lundberg & Dorr at Deadwood. About 10 years later the firm of Lundberg, Dorr & Wilson was organized and bought the property, believing that it still contained a large amount of $4 to $6 ore that could be made to yield a profit by treatment in a modern mill at the mines. Accordingly construction of the present mill and tramway was begun in 1903 and operations commenced in January, 1904.

The estimated mill capacity of 75 tons per day was gradually increased to 110 tons; and combined mining and milling costs were reduced until ore averaging $3 per ton could be profitably handled.

The mill is of interest historically as having passed through the whole period of American cyanide practice. When designed, leaching was the standard method of treatment. Slime was the *bête noir* of the millman, both on account of the difficulty of separation from sand and the recognized loss in decantation, which was the only method of slime treatment in use in this country.

Development of Sand-Slime Separation

The ore handled at this mill produced a great deal of slime, with the attendant difficulties in treatment. A system of double cone classification was originally installed, but failed utterly to make a satisfactory separation of sand and slime. The frequent changes in the nature of the ore aggravated the conditions and it was impossible to produce a leachable sand or a slime free from coarse sand. The constant attention demanded by the cones added also to the expense of operation.

The difficulties thus encountered were finally overcome by the invention of the Dorr mechanical classifier in July, 1904, and this machine proved responsible for the ultimate success of the mill. The original classifier was replaced in 1908 by a Model B machine, and the latter was in continuous operation from that time until the mill closed. The total cost for repairs during that period was not over $5, and the classifier is apparently in condition to run another five years.

The use of this machine and of other mechanical classifiers has since spread over the whole country and into most of the cyaniding districts of the world.

Another item of interest is the fact that this mill was the pioneer in the Black Hills in the use of electric power purchased from a central plant. It was also the first in the district to use Chilean mills for fine grinding and belt elevators for raising pulp ground in cyanide solution. Finally it had the distinction of using the first vacuum leaf-filter which remained in successful operation for any length of time.

The total quantity of ore milled was approximately 285,000 tons, of which about three-fifths came from the firm's mines and the balance from custom ore. Owing to the operation of the mill as a custom plant, no figures on operating costs or extraction have been published.

The total production has approximated $1,500,000 and the net earnings, after deducting the cost of the property and all improvements, have been several times as great as estimated when the property was bought, although the actual value per ton of ore proved to be less than originally estimated.

Coarse Crushing and Sampling

Company ore was delivered by surface and aerial tramways to a 135-ton bin at the top of the mill. The quantity of ore thus received was estimated by periodically weighing the mine cars and applying average weights to the number of loads delivered. Custom ore was received by railroad and accurately weighed on track scales. It was then dumped into one of two bins of 40 and 60 tons capacity, situated under the tracks about 80 ft. from the mill, and trammed by hand to the crusher.

All ore was passed over grizzlies of 1½-in. space, and the over-size was crushed in a No. 4D Gates gyratory. The Gates product joined the grizzly undersize and was elevated to the crushed-ore bin of 75 tons capacity. The elevator was 37 ft. high, with 12-in. belt and No. 8 Acme buckets, 12 in. by 6 in. by 5 in.

An automatic sampler was placed at the elevator discharge. This machine was of local construction, designed by Mr. Dorr. It consisted of a narrow spout which was drawn across the discharge chute at regular intervals, cutting out a portion of the ore stream and delivering it to the crusher floor, where it was further crushed and sampled for assay. The sampler was operated by a stream of solution flowing into a counterbalanced box placed in the main mill building.

Accurate sampling was a necessity at this mill, for during the entire period of its operation the company purchased and treated over $1,000,000 worth of ore. Figures for the period

1907-1912, inclusive, show that the average value of all ore treated was $5.11 per ton, and during this time there was a variation of but 1 cent per ton between the value indicated by the automatic sample and another taken from the feed to the Chilean mill.

Grinding in Cyanide Solution

From the crushed-ore bin the ore was fed by a shaking-trough feeder to a set of 14-in. by 30-in. Carterville geared rolls. A spray of cyanide solution was applied from above to keep the rolls clean and the crushed ore was sluiced to the Chilean mill by an additional stream of solution. Operated in this way, wet roll-crushing proved entirely satisfactory. Fig. 2 gives a view of the rolls and Chilean mill.

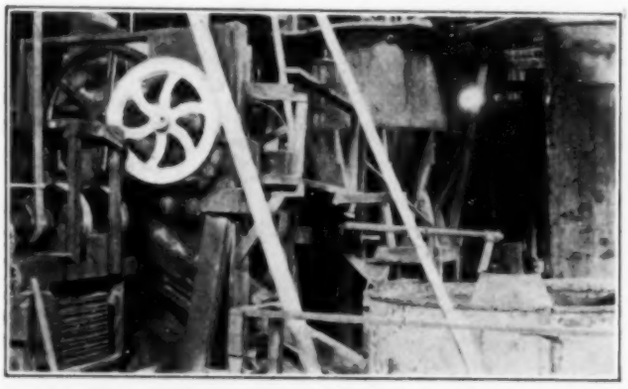


FIG. 2.—GEARED ROLLS, CHILEAN MILL AND ELEVATOR

The Chilean mill was a 6-ft. Trent Monadnock, fed through a launder entering the housing about 6 in. above the screens. This method of feeding was adopted several years ago on information received from Mr. Orford, of Delamar, Nevada. It causes no irregular wear of the die and reduces wear on the feed launder supplied with the mill.

This Chilean mill made a remarkable operating record. The bronze spindle bushing supplied with the mill was still in use when the plant was closed. The original gear wheel was changed two years ago after six years' service, when the original spindle was replaced with another of improved type, designed at the Mogul mill, at Pluma.

Additional cyanide solution is fed to the Chilean to make the ratio of solution to ore about 5:1. Tyler ton-cap screens of different mesh have been used with satisfaction. The following tabulation gives screen analyses of the ground pulp for two different screens and ratios of solution to solids:

Mesh.	Ton-cap screen 0.025-in. slot. 4.53 tons solution per ton ore.	Ton-cap screen 0.020-in. slot. 6.66 tons solution per ton ore.
+ 20	0.60%	0.00%
+ 30	8.22	4.72
+ 40	7.38	5.12
+ 60	16.82	15.05
+100	18.24	18.30
+200	15.50	21.66
—200	32.94	34.25

The metal consumed in the various grinding machines, per ton of ore treated, is shown in the following table. The figures are based on the assumption that the entire metal is used:

Machine.	Lb. Metal per ton ore.
Rolls	.0902
Chilean mill	
Tires	.4046
Dies	.3024
Gyratory	
Concaves	.0012
Head	.0357

The average life of the different crusher parts was: Roll shells, 10 mo.; Chilean tires, 4 mo.; dies, 3 mo.; gyratory concaves, 16 mo.; heads, 12 mo.

Sand-Slime Classification

The Chilean mill product was elevated by belt and bucket elevator fitted with No. 14 Salem buckets, 12 in. by 6 in. by 5 in. Only two sets of these buckets were used during eight years of operation. Belts lasted about two years. The elevator housing was of canvas, protected with an inner lining of steel at points of excessive wear. The elevator discharged into a Dorr duplex Model B classifier, the sand product of which passed to leaching vats and the slime to cones and a Dorr thickener. The separation produced about 45 per cent sand and 55 per cent slime, the following being the exact figures for the past four years:

	1909	1910	1911	1912
Per cent sand.....	45.0	48.6	52.0	47.2
Per cent slime.....	55.0	51.3	47.9	52.8

Sand Leaching

The sand from the classifiers was sluiced with solution, through launders having a pitch of 1½ in. to the foot, to the leaching vats. A view of this part of the mill is given in Fig. 3, which shows the 8-arm, ball-bearing wooden distributor which served the line of vats. The latter were 18 ft. diameter and 11 ft. deep. Their original capacity was increased to 106 tons by the simple expedient of an inside launder raised 12 in. above the rim, as is shown clearly in the figure. During the early operation of the mill the sand vats were discharged by hand through bottom gates, and the dry sand was trammed to the dump in 4-ton cars of the Mercur type. The cost of this operation was $6 per tank. Later the system was changed and the vats were adapted to discharge by sluicing.

The total time allowed for sand treatment was five days, of which 32 hours was required for filling a vat, 12 hours for leaching with solution and the balance for washing with barren solution and water.

Vacuum Filtration for Slime

Two settling cones, one 18 ft. and the other 22 ft. diameter, were formerly used for thickening slime for filtration. They were so placed in the mill that the clear, overflowing solution would gravitate to the grinding apparatus. The usual difficulty was experienced with these cones, viz., that solid slime would build up on the sides, making it impossible to secure a constant underflow of uniform consistency. Dissatisfaction with these devices led to the invention of the Dorr continuous thickener at the Mogul mill in Pluma, and when more settling capacity was required at the mill under consideration, one of the sand vats was equipped with the thickening mechanism and proved of great benefit. This thickener is 18 ft. diameter and 8 ft. deep, and has a capacity of from 40 to 50 tons in 24 hours. It was operated at 1/10 r.p.m., and actual power meas-

FIG. 3.—SAND LEACHING VATS AND DISTRIBUTOR

urements showed a consumption of only 1/20 hp at the pulley.

The system of slime treatment was very simple. The slime from the classifier was divided between the cones and thickener. The clear overflow from the former gravitated to the roll-supply tank and from the latter to the Chilean mill. Thickened slime from the Dorr machine and from the 18-ft. cone was pumped by air-lift to the 22-ft. cone, and thence to an 8-ft. air-agitation cone which discharged to the loading tank of the Moore filter.

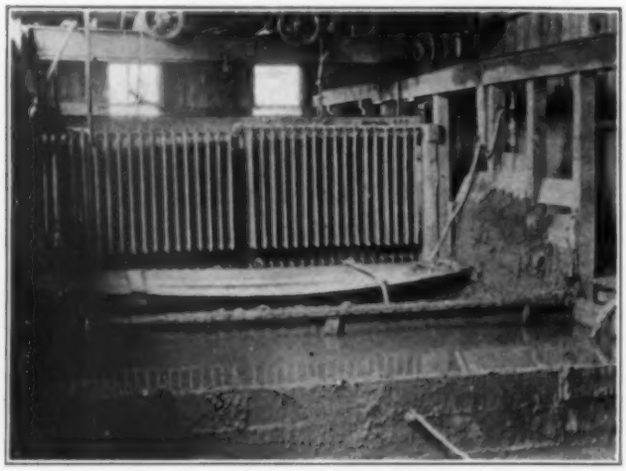


FIG. 4.—MOORE VACUUM FILTER

The filter equipment comprised five tanks, one each for loading, barren solution wash, water wash, discharging and acid treating; two sets of 34 filter leaves, 6 ft. 6 in. by 4 ft. 6 in.; and a 10-ton hydraulic crane operated under pressure of 105 lb. The loading and discharge tanks had hopper bottoms. The cycle of filtration varied with the condition of the leaves. The loading time was 25 to 35 minutes. Barren solution wash was applied for 45 to 60 minutes and wash water for 25 minutes. Ten minutes was required for discharge of cake and return of filter to the loading tank. The moisture in the feed was usually maintained at from 60 per cent to 70 per cent, and the discharged cake contained 25 per cent to 35 per cent moisture. A view of the filter is given in Fig. 4.

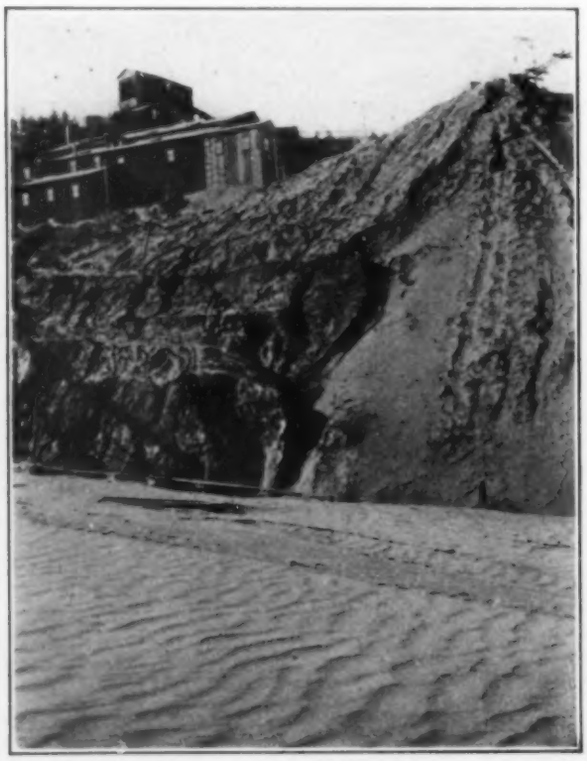


FIG. 5.—SAND TAILINGS IN FOREGROUND, STACKED SLIME TAILINGS IN BACKGROUND

Until two years ago the filtered slime was trammed from the mill and spread over the hillside. About that time the creek was piped for some distance along the foot of the dump and it became possible to sluice tailings to the bottom of the gulch. The early necessity of discharging nearly dry tailings and holding them on the side of the gulch, as well as the anticipated efficiency in slime filtration, led to the adoption of the Moore filter process. By not discharging the cake under water, it has been possible to produce tailings dry enough to be retained on the dump without building walls or dams. The view given in Fig. 5 shows sand tailings in the foreground and slime tailings banked in the background.

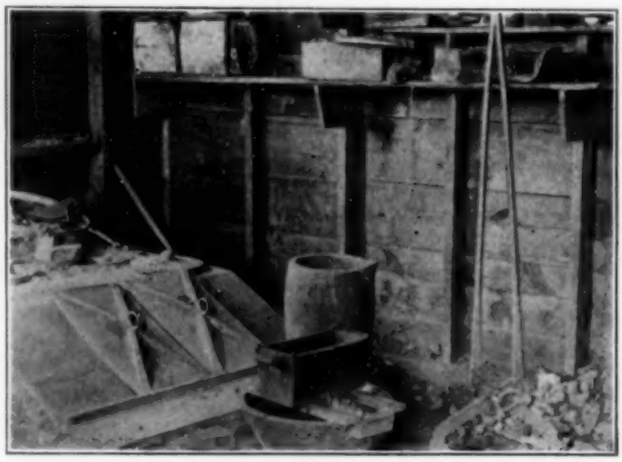


FIG. 6.—COKE FURNACE FOR MELTING BULLION

The mechanical loss of cyanide was determined by a very careful series of tests to be 0.3 lb. when washing with a 1-lb. barren solution.

Precipitation and Refining

Gold-bearing solution from the slime filter and the sand-leaching vats was precipitated on zinc shavings. The latter was packed in 30 barrels, arranged in 10 sets of 3 each, having a total capacity of 60 cu. ft. of shavings. Precipitation has been entirely satisfactory, the gold value of the tailing solutions ranging from 2 to 6 cents per ton.

It was the usual practice to make a clean-up twice a month, at which time all the contents of the head barrel and part of the second and third barrels were removed. The coarse zinc was sorted and returned to the second barrel and the fine zinc and gold precipitate refined. The barrels were then moved up one step, the second being placed at the head and the first at the foot of the series. A trolley was used for handling the barrels. The gold precipitate and fine zinc was treated with sulphuric acid, washed, filtered and dried in a shelf furnace, where it was exposed to the gases of combustion. The dried precipitate was then fluxed and melted in the coke furnace shown in Fig. 6.

One reason for the success of the property has been the indefatigable energy and careful attention of Mr. J. M. Pennington, Jr., the manager. Mr. Pennington entered the employ of the firm as assayer at the beginning of operations and gradually assumed more responsibility until he became manager. Many of the men employed in the mines and mill, including the shiftmen, were with the firm during nearly the whole time of operation, and by their faithful and intelligent service contributed materially to the success of the venture.

The Arizona Copper Co. produced only 1300 tons of copper during July, the output being interfered with by the work of enlarging No. 6 concentrator.

An Electrical Method for Comparing Thermal Conductivities

By Wm. Eves, 3rd

Most methods for measuring or comparing thermal conductivities require the determination accurately of the working temperatures of the source of heat and of the sink by which the heat is carried away. Although there are many very accurate ways of determining temperatures by electrical means and otherwise, yet in the particular case of heat flow, conditions arise which cause errors in obtaining the actual temperatures of the points required.

The method for comparing thermal conductivities, which is described in this article, was suggested and devised by Dr. E. F. Northrup, who greatly assisted the writer in carrying it out. It does not give thermal conductivity in absolute measure, but by means of it, one substance may be compared with a standard substance of which the thermal conductivity is known.

The method was first used for comparing the thermal conductivities of commercial impregnating compounds such as are used for the field coils of electrical machines. The operation of this measurement is described in this article. It is now being extended to the comparison of any solid substances, a report of which will be given later.

Briefly, this experiment consists in the heating of two coils of copper wire, identical in every respect, namely, size, shape, electrical resistance and in insulation, except for the compounds with which they are impregnated and of which the conductivities are to be compared. This heating is done by passing such currents through them that will cause them to become each equal in resistance to some set resistance, which is previously calculated and the magnitude of which determines the mean temperature of the coils. Sufficient time must be taken to allow a steady state of heat flow to be reached, in order not to obtain false results.

If this steady state has been reached and if the external temperatures are maintained equal, the total energy input into each coil will be equal to the heat flow and therefore to the energy dissipated by the coil. Since the coils are identical in size and shape, the thermal conductivities of the wire, insulation and impregnating compounds of the coils will be proportional to the amount of energy input, since the material of greater conductivity allows proportionately greater energy to be conducted through it. The watts dissipated from the two coils will be:

$$\text{Coil A:}\quad W_1 = C_1^2 R_t \tag{1}$$

$$\text{Coil B:}\quad W_2 = C_2^2 R_t \tag{2}$$

where R_t equals the resistance of coils A and B at the higher temperature t, and C_1 and C_2 are currents passing through A and B, respectively.

Therefore,

$$\frac{k_A}{k_B} = \frac{W_1}{W_2} = \frac{C_1^2 R_t}{C_2^2 R_t} = \frac{C_1^2}{C_2^2} \tag{3}$$

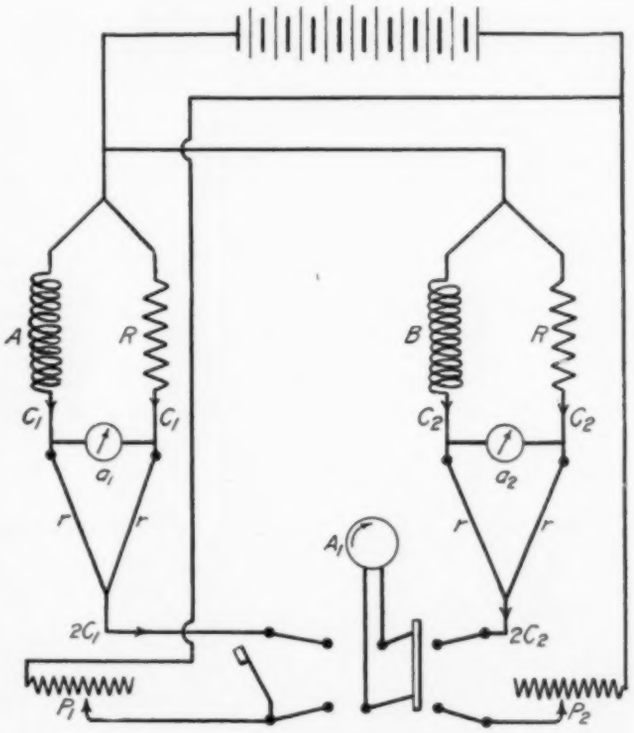
where k_A = conducticity of coil A and k_B = conductivity of coil B. Hence it follows that for equal mean internal temperatures steadily maintained for both coils, their thermal conductivities will be in direct ratio to the squares of the currents passed through each, which are measured as described below.

Four identical coils of No. 20 double cotton-covered copper wire were wound on a form so that they were 10 cm in diameter and 10 cm long in order to obtain minimum surface for a given weight. They were bound with tape to keep from falling apart and the forms taken off, leaving a 1-cm hole running through the center. Three of these were then impregnated with three different commercial insulating compounds, which are called A, B, C and the fourth coil D was left unimpregnated. These coils were adjusted so that their resistances were equal to 1/5 of 1 per cent at the same temperature, and the two to be compared were placed in the apparatus as shown by the coils A and B in Fig. 1.

In main, the apparatus consists of two Wheatstone bridges, in which the heating current passes through the bridges. Coils R,R are adjustable resistances of wire of very low and equal, or zero, temperature coefficient, which are set to the calculated resistance which the coils A and B will have at the temperature at which it is desired to take the measurement.

Resistances r, r, r, r are ratio resistances, equal in value. In this case they were made of nichrome strip mounted on a board with binding posts for terminals. They should be low is resistance and with large area in order to prevent heating to excess. They were also adjusted to within 1/5 of 1 per cent and were taken equal approximately to 1/10 of the coils R,R.

a_1 and a_2 are small portable galvanometers, which were found sufficiently sensitive for this method. In fact millivoltmeters could be used. A is an ammeter connected through a double-throw switch and short-circuiting switches, and of suitable range to measure the total currents passing through the



METHOD FOR COMPARING CONDUCTIVITIES

bridges, which are each twice the currents flowing through the coils A and B, or R,R, since the arms of the bridges are equal in resistance. p_1 *and* p_2 are variable rheostats of any rough type for adjusting the impressed voltage.

Current was passed through the bridges until the detectors showed that the mean internal temperatures of the coils A and B had risen until their resistances were equal to the set resistances R,R. Thus the internal temperatures of A and B are identical provided the copper in each has the same temperature coefficient. But if they should differ in temperature coefficient, coils R and R can be adjusted to the proper resistances by knowing the coefficients, so that they will come to the same temperature when the instruments a_1 and a_2 show balances in the bridges.

The coils A and B have the same external temperature provided the proper precautions have been taken, such as a constant temperature bath well stirred. But in this case the coils were placed very near together in the air, so as to more nearly represent actual conditions on an electrical machine and still be certain of having the same external temperature. The experiment was carried out in a closed room, free from drafts,

so that this steady state was quite easily maintained, as indicated by the detectors.

Coils B and C were each compared with coil A, in order to obtain comparisons of the impregnating compounds, and also coils A and D were compared, in order to compare the ability of an unimpregnated coil to get rid of its heat with the ability of an impregnated coil, and the results showed that the dead air pockets between the wire layers are relatively poor heat conductors. In each case, thermometers were placed in the central holes of the coils, merely as a check. One was also placed about half a meter away from the coil to determine the external or room temperature.

These temperatures were not at all necessary except to have some idea as to the temperature difference between inside and out, since the experiment was continued by making the measurements at various differences of temperatures by raising the internal temperature. It was found that the ratio of the conductivities did not vary within the limits of experimental error for different differences of temperatures, although it has been shown by other experimenters that in wider ranges the conductivity varies.

The numerical results of the measurements are as follows:

Resistances of coils A, B, C and D at 23.7 deg. C. = 8.91 ohms.

Run No. 1, Feb. 28, 1913. Observer W. E. Comparison of coils A and B.

$R = 8.91\ [1 + 0.0038\ (80 - 23.7)] = 10.806$ ohms.

$r = 1.156$ ohms

Current coil A, $2C_1 = 2.223$ amps. Current coil B, $2C_2 = 2.098$ amps.

Mean internal temperature = 80.0 deg. C.

Observed room temperature = 23.9 deg. C.

Conclusion: $k_A : k_B = 112.3 : 100$.

Run No. 2, March 12, 1913. Observer W. E. Comparison of coils A and D (coil D unimpregnated).

$R = 10.806$ ohms.

$r = 1.150$ ohms.

Current coil A, $2\ C_1 = 2.268$ amp. Current coil D, $2C_2 = 1.880$ amps.

Mean internal temperature = 80.0 deg. C.

Observed room temperature = 23.9 deg. C.

Conclusion: $k_A : k_D = 145.5 : 100$.

Run No. 3, March 15, 1913. Observer W. E. Comparison of coils A and D (temperature of coils higher than No. 2).

$R = 8.91\ [1 + 0.0038\ (105.7 - 23.7)] = 11.619$ ohms.

$r = 1.150$ ohms.

Current coil A, $2C_1 = 2.715$ amps. Current coil D, $2C_2 = 2.228$ amps.

Mean internal temperature = 105.7 deg. C.

Observed room temperature = 23.5 deg. C.

Conclusion: $k_A : k_D = 148.5 : 100$.

N. B.—During this run the resistance of coil D increased to 8.98 ohms at 25.3 C., causing lower current value for coil D, and therefore a greater ratio of conductivities than is correct.

Run No. 4, March 29, 1913. Observer W. E. Comparison of coils A and D (at still higher temperatures than No. 3).

$R = 8.91\ [1 + 0.0038\ (134 - 23.7)] = 12.634$ ohms.

$r = 1.148$ ohms.

Current coil A, $2C_1 = 3.124$ amps. Current coil D, $2C_2 = 2.583$ amps.

Mean internal temperature = 134.0 deg. C.

Observed room temperature = 25.0 deg. C.

Conclusion: $k_A : k_D = 146.3 : 100$.

Run No. 5, April 1, 1913. Observer W. E. Comparison of coils B and C:

$R = 10.806$ ohms.

$r = 1.105$ ohms.

Current coil B, $2C_1 = 2.102$ amps. Current coil C, $2C_2 = 2.263$ amps.

Mean internal temperature = 80.0 deg. C.

Observed room temperature = 23.5 deg. C.

Conclusion: $k_C : k_B = 115.9 : 100$.

Summary:

Comparison of coils A and D at different temperatures:

$k_A : k_D$ at 80.0°—23.9° = 145.5 : 100.

$k_A : k_D$ at 105.7°—23.5° = 148.5 : 100 (see note Run No. 3.)

$k_A : k_D$ at 134.0°—23.7° = 146.3 : 100.

Comparisons of coils A, B, C and D at 80.0°—23.5° and 23.9°:

$k_C : k_A : k_B : k_D = 100 : 96.9 : 86.3 : 66.6$.

The resistances were always measured before and after each run and in no case, except the run noted was the error greater than 0.4 of 1 per cent. Briefly, the observations show that the ratio of the conductivities changed little if any in the range of temperature used. Two of the compounds tested were far superior to the third and all three had much higher conductivities than an unimpregnated coil, showing that the impregnating compounds tested will not only make a better insulation, but will conduct away the heat faster than trapped air.

The most necessary precaution is to continue the experiment for several hours after an apparently steady state has been reached. In most cases, an observation can be completed in twenty-four hours if readings are taken and the current values changed once an hour or probably more often. A change of one-tenth degree in the difference between internal and external temperatures, caused approximately a change of one scale division of the galvanometer, but such accuracy in temperature is not claimed. Nevertheless, it shows the sensitivity of the method. The accuracy desired at the beginning of the experiment was within 1 per cent., and it is safe to assume that this was obtained, since all resistances checked better than to 0.4 of 1 per cent., and the current could be read to better than 0.4 of 1 per cent. The source of e.m.f. used was from 24 to 36 volts direct current, and by means of a flexible system of rheostats the adjustments and measurements are very easily made. The apparatus may be set up in a small space and may be made up entirely of commercial apparatus, the accuracy of which, of course, should be checked, as was done with all instruments in this experiment. The method is commercially practical, in that a measurement may be very quickly made after a short experience and with a little foresight, and also since it is not necessary to make any corrections whatsoever due to theoretical inaccuracies.

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Princeton, N. J.

Electric Furnaces, Their Design, Characteristics and Commercial Application

By Woolsey McA. Johnson and George N. Sieger

In presenting this series of articles to the public, the authors wish to state their three-fold aim as being to give in concise form the scientific, engineering and commercial principles and facts involved in the industrial utilization of electric furnaces. The serial embodying this idea is designed for three classes of readers, to wit:

1. Engineering students who wish to learn the technology of electric furnaces.

2. Engineers who are desirous of having in convenient form a "desk-book" covering the latest intensive application of electrical energy.

3. Business men who might need to learn quickly for a specific purpose the general principles of electric furnaces.

As the business principles of any applied science are in reality the most important and oftenest neglected, the third aspect has been emphasized everywhere, in order that the authors and the readers might have conjointly in mind the relation between the dollar and the kilowatt hour.

A short ton (a long ton being practically equal to a metric ton) of average coal with a heating value of 11,600 b.t.u. per pound contains exactly the same heat as a horsepower year. The ton of coal costs from $1.00 to $8.00 with an average value f.o.b. factory for the country of $2.00. A horsepower year costs from $10.00 to $50.00, with an estimated average value at the American electro-metallurgical plants of $18.00. It is seen clearly the cost per unit of heat, e.g. b.t.u. or caloric, is greatly in favor of coal or other fuel as gas or oil and that

electric heat is only applicable where certain metallurgical peculiar advantages exist or where its application is so convenient and so efficient in other ways that its primal disadvantage of large cost per calorie is overcome by other inherent but superior characteristics.

With this thought held prominently in mind, the scientific facts and practical details become aids and props and without it the commercial advance is greatly hindered. While much that is apparently useless to the practical man will be taken up, sooner or later he will appreciate its value and the scientific and commercial facts which he has mastered will yield him an increased profit.

In order to have a dividing line between electric furnaces and other classes of electric heating apparatus such as electrically heated ovens, etc., an arbitrary line has been drawn and the authors do not consider as an electric furnace any electrical heating apparatus where the working temperature is below the minimum visual red, e.g. 550 deg. C. or 1022 deg. F.

Article I—General Principles of Electric Furnaces Design and Economics

The commercial application and the cost of heat of course govern the industrial progress of metallurgy. It is a fact that electric heat is expensive and to be used at all must be used efficiently.

The essential differences between heating with carbonaceous fuel and electrically where there is no radiation or heat lost (there are never any furnaces where there is no radiation, but for purposes of illustration such ideal furnaces are a convenient assumption) are expressed as follows:

a. Flow of heat from where it is generated to where it is usefully absorbed is a function of the temperature difference and a constant of resistance, and a law analagous to Ohm's law obtains.

$$1.\quad H\ (\text{Calories transmitted in unit of time}) = \frac{T_s - T_{ab}}{K_r}$$

Where K_r is the constant of thermal resistance for any given furnace, T_s temperature of source of heat and T_{ab} temperature of absorption or the furnace's working temperature. K_r will vary with size and type of furnace.

b. In "carbon-heated" furnaces T_g has a definite upper limit because not enough heat is generated to do more than raise the products of combustion above a certain maximum temperature, e.g., Richard's Metallurgical Calculations, part I, page 36 *et seq.*

c. Where T_{ab} or working temperature is high, *e. g.*, in a ferrosilicon furnace, $T_g - T_{ab}$ must be small. K_r is a constant, hence heat imparted usefully is small because the temperature difference between source of heat and place of heat absorption is small. Finally when this difference becomes zero no heat is usefully transmitted and the operation becomes uncommercial. But the temperature difference is the driving force.

d. K_r the constant of thermal resistance of any given furnace depends on its size and shape. These are governed by the design, and the design limited by nature of reactions proceeding. For instance, in an induction furnace K_r is extremely small, for the heat is generated directly in the metal bath, whereas in a reverberatory arc or resistance type as in De Laval zinc furnace, the heat must be reflected down and transferred by radiation, convection or conduction. Hence, K_r is in this case large, and to attain a high thermal efficiency T_s must be high, 2500 deg. C. plus, and only limited by danger of melting down arch.

Let us now consider, in rough approximation, the limits of carbon-heated furnaces and electric furnaces when there is no radiation. No mathematical analysis is needed because the principles stand out in bold simplicity. These principles are:

(1.) The heat generated and imparted per cubic foot of furnace laboratory (working cubical contents of furnace) is several times as great in the electric furnace as in a "carbon-heated" furnace.

(2.) The percentage of heat wasted in radiation and conduction through the furnace walls therefore must be much less in an electric furnace than in a carbon-heated furnace and hence entirely apart from consideration of equation 1 electric furnaces are usually much more efficient thermally considered than their rivals. Moreover, the design of the shape of walls and materials used often allows better heat insulation.

When we consider the fact that radiation is always much less and sometimes several times smaller, the thermal efficiency of the electric furnace is many times higher.

Returning to mathematical expression, equation 1 can be transformed thus:

$$2.\quad H\ (\text{Cal. transmitted per unit of time}) = \frac{T_s - T_{ab}}{K_r} - R$$

where R equals heat lost by radiation to the surroundings of the furnace.

$$3.\quad \text{Thermal Efficiency} = \frac{\dfrac{T_s - T_{ab}}{K_r} - R}{H}$$

A consideration of equations 1, 2 and 3 shows that in certain cases thermal efficiency becomes quickly zero and it would be unprofitable to attempt to perform any thermal or thermochemical work above the normal limits of T_g, which are fixed strictly by the laws of chemistry and physics at 1600 deg. C. to 1900 deg. C. in the cases of most furnaces heated by fuel or, to use a new term which we will adopt, "carbon-heated" furnaces.

For instance, in the manufacture of ferrosilicon the iron blast furnace is now more economical, making a product of 80 per cent Fe, 20 per cent Si, but the economical thermal limit on which is superimposed the metallurgical limits of the need of an intensely reducing atmosphere (much easier attainable in an electric furnace where no air is blown than in a blast furnace where air *must* be blown in to burn up the coke and thus make heat). But above this percentage the blast furnace does not compete with the electric furnace and all ferrosilicon about 20 per cent Si is made today in electric furnaces. The electric furnace competing with itself makes ferrosilicon in most places most economically of 60 per cent Si and 40 per cent Fe.

A study of the theoretical temperatures necessary in the manufacture of ferrosilicon and the theoretical temperatures possible with a coke furnace supplied with hot blast indicates that as T_{ab} increases with an increase of percentage of silicon in product, the quantity $T_s - T_{ab}$ becomes smaller or the direct thermal "drive" is less, for T_s is fixed by laws of thermochemistry and physics. It is thus costing much more to transfer heat to the charge, we find that as the percentage of silicon increases electricity is a better commercial agent than its rival, because the temperature required for reaction is higher.

When the electric furnace begins to compete with itself, we find that as the requisite working temperature increases, the radiation from the furnace is so increased that a large amount of money is lost in heat dissipated to the atmosphere. So, it is better business to make a ferrosilicon high enough in silicon to stand a shipment but not so high that kw-hours per lb. of silicon is a large number. With cheap power it would be good business to make a richer ferrosilicon, other things always being equal.

Let us reiterate that as the cost of an electrical calorie is three to ten times greater than the cost of a "carbon" calorie, electric heat is only commercially applicable because of its efficiency in other lines.

With this in mind, in the comparison between the electric furnaces and their rivals and taking into consideration the design of electric furnaces in light of this comparison, it becomes self-evident that there are certain inherent chemical or, better expressed, metallurgical and commercial points of superiority of the former over the latter.

In any case where strongly reducing conditions above 800 deg. C. are needed the electric furnace begins to become the better to use, and as the temperature requisite becomes higher their inherent chemical superiority is increased. The *commercial* inherent superiority is also increased because such heating under reducing conditions must be done indirectly through the walls of a muffle, retort, or crucible, and as the temperature is increased the size of the unit is decreased. Hence the labor charge is greatly increased and labor-saving devices so necessary to-day cannot be used. Such conditions exist in the zinc retorting plant, the brass foundry, the crucible steel plant, and the gradual commercial elimination of these three is inevitable.

As we go higher in the scale of reductivity and temperature we see the calcium carbide, the carborundum, the alundum, and the graphite furnace where no competition by coke-fired furnaces can exist, for the reactions both chemical and physical are scientifically impossible as well as commercially impossible save in an electric furnace. By a natural industrial evolution these operations have arrived as commercial successes quicker and in the main easier than in the other cases above mentioned.

The following facts can therefore be laid down as true:

(1) The relative thermal efficiency of carbon and electric furnaces are such that the higher the temperature and the more difficult the metallurgical work to be done, the more advantageous is the electric furnace.*

(2) The relative commercial efficiencies of the carbon-fired and electric furnaces are such that the harder the metallurgical work to be done the greater the inherent points of superiority of the electric furnace.

As the design and characteristics of electric furnaces become better understood, other advantages, it can be prophesied, will be more prominent.

These are:

(1) Control of temperature.

(2) Control over quality of product.

(3) Decreased cost of repairs and renewals.

(4) Adoption of machinery for handling raw materials and products.

(5) Decreased capital cost because of intensity of heat transfer.

(6) A more efficient use of high-priced intelligent labor.

These six sub-advantages must always be considered in the general design of electric furnaces, even though their existence is not so apparent.

In general, it can be stated as a fact that the closer the source of heat is to the place of the absorption of the heat by the charge, provided there is a regular flow of electrical energy and charge, the closer will be the control over temperature and over quality of product. Thus, we are also making thermal efficiency higher by making K_r smaller.

For this reason a number of smaller electrodes is better than a few large ones. Large thick walls and a big bath of slag or metal act as thermal balance wheels and keep conditions more uniform. Increase of size of unit, provided the characteristics of the furnace and process are really understood, also make for all of the above sub-advantages.

Some Practical Types of Electric Furnaces

Unquestionably there is no form of electric furnace simpler than the "DIRECT RESISTANCE" type invented by A. H. & E. A. Cowles in 1879. This as used by Acheson in his carborundum and graphite furnace is simple, practical and efficient. The disadvantage of building a new furnace after each run, the way the Chinese ancestrally have done in iron smelting, is of course apparent, but the fact that there is a monopolistic market for the product renders it unnecessary to make metallurgical and mechanical refinements. The commercial aim is to turn out the goods and sell them. Salesmanship is the essential thing after the goods are made.

A much more finished metallurgical instrument is the "ARC FURNACE OF THE SIEMENS TYPE." This as developed by Heroult and engineers of the United States Steel Corporation, by Girod and others for making a high quality of steel on a tonnage basis with automatic feed of electrodes in a tilting furnace, is pretty near a finished invention.

The "REVERBERATORY" ARC FURNACE where the arc passes from one upper electrode to another and as developed by Stassano for steel, de Laval and Weeks for zinc dross and zinc ores, has a much narrower field than was supposed recently. The fact that the heat is generated some distance from the charge and reflected down from the arch makes the transfer of heat slow and costly. Moreover, the danger of melting down the arch, common in open-hearth steel furnaces and copper reverberatories, is multiplied several times in this type of a furnace.

The "BURIED ARC" type as used for the manufacture of calcium carbide or the slag resistance type, for the two blend into each other, such as used by Irvine for the production of phosphorus is a furnace capable of wide metallurgical application. Both the two main advantages and the six sub-advantages are present in natural and balanced proportions. Rightly applied and designed for continuous work so that each part of the furnace is functionizing at a high efficiency and delivering its charge to parts below it with charge progressively operated upon, these types of furnaces have great commercial possibilities.

Electrical Heat Must Be Applied Efficiently

Considered in another aspect, we have the subject of preheating by heat generated by carbonaceous fuel. As has been shown, the imparting of thermal energy by electrical means to a charge becomes commercially much more efficient as the temperature increases. Thus it is good common sense and sound business to do where possible low-temperature heating by the heat of combustion of carbonaceous fuel until its efficiency becomes low and the high-temperature work by electrically generated heat. Col. Robert M. Thompson, of the International Nickel Company, former employer of one of the authors, enunciated this principle of economic electrometallurgy: *"Electricity is an expensive agent and should only be used for the finishing metallurgical operation."* Applying this principle directly, it can be stated axiomatically that in any metallurgical process, especially that of a commercial nature, the first work to be done should be done by fuel-generated heat and the last work to be done should be done by electrically-generated heat.

Broadly speaking, the electric furnace process should be so designed that each co-ordinate part does its duty as cheaply as possible and likewise should be considered both as it attains the end and also in relation to the other parts.

Mr. J. E. Johnson, Jr., has analyzed the different functionating parts of the iron blast furnace, the chemical work to be done and the chemical force applied from the tuyeres to the charging floor. He has found that the design of the modern American blast-furnace as worked out by an evolutionary process conforms to the law that there should be a progressive application of metallurgical agents each doing its work at a maximum efficiency.

This principle, which is correspondent with the "Thompson principle," should be applied in the design of an electric furnace and in electric furnace processes.

Thus we find that too large electrodes and the resultant generation of a large amount of heat will often produce "hot spots" in an electric furnace and that this localized and intensified electric heating will produce untoward effects such as burning out the lining of the furnace in proximity of the electrodes or the reduction of oxides to their elemental forms, with the subsequent vaporizing of, for instance, silicon and aluminium.

While for obvious practical reasons it is well to increase the kilowatts generated per cubic foot of working space to a maximum, yet this maximum should be within safe workable limits and no overheating either of the entire furnace or in spots is permissible. The kilowatts generated per cubic foot of

* Note it can be estimated that in general to-day the dividing line is about 1100° C. (2000° F.) with a margin of 200 or 300° C. each way.

working space must then be placed at a safe maximum as allowed by other metallurgical and superimposed particular conditions.

High Voltage a Good Commercial Point

As will be shown in a later article, the power factor of an electric furnace becomes lower as the size of the unit increases and above 1000 kw the inductive drop becomes considerable. Since the amount of copper tied up in the form of bus-bars, switches, transformers and generating plant is so increased without financial return, the furnace should be designed to be operated at as high a voltage as possible, for a high-voltage design will increase power factor and commercial efficiency in other obvious ways.

So too with the "load factor" (the actual output of a station in kw-hours during a given period divided by the output obtainable by operating station at its full rated capacity for that period), for the best power plant operates somewhat irregularly. The electric furnace should if possible be so designed as to absorb the fluctuations of the output of power plant, usefully. By shape of unit, contents and thermal insulation thereof, it is possible to have a number of electric furnace units that will give to a power plant a "load factor" that would be, practically speaking, nearly 100 per cent.

The question of heat insulation is not nearly so important as the layman deems it to be. By increasing to 5 or 10 kw per cubic foot of laboratory it is possible to design even as small a unit as 200 kw with water-cooled walls emitting 0.5 kw per square foot of free surface with a thermal efficiency of 60 to 70 per cent.

Where a corrosive slag is used these walls will have a long life, whereas walls well insulated have a short, uncertain and a merry life.

Far more important is the geographical location of the plant, the kw-hours per pound of product, and the cost of power per kw-hour. The cost of power per kw-hour and the cost of power per pound of product are positive determining factors in the success of any electric furnace process. With the immense amounts of secondary hydroelectric power available at less than 2.5 mills per kw-hour for nine or ten months in the year, it is of prime commercial importance that electric furnaces and companion processes be developed that absorb in a manner useful to the nation this waste power. Undoubtedly this application of the electric furnace will be a larger one in the future.

Summary

This chapter can be summarized as follows:

1. Electric heat is expensive and can only be used where its efficiency is great.

2. Electric heat can be employed in certain work where it is needed badly.

3. If intelligently studied and applied, electric furnacing will become of large industrial importance and the existing limits pushed to extremes not now thought possible.

If an electric furnace be thought of as a short-circuit under control, we must strive to put it under good and intelligent control.

Notes on Chemistry and Metallurgy in Great Britain

(By our Special Correspondent)

Institution of Mechanical Engineers

The summer conference of the Institution of Mechanical Engineers was held at Cambridge. Sir H. F. Donaldson presided and there was a good attendance of well-known members. The two papers taken on the first day were "A New Method of Cooling Gas Engines," by Professor **Bertram Hopkinson**, and "Modern Methods of Measuring Temperature," by Mr. **Robert S. Whipple.**

The Cooling of Gas Engines

Professor Hopkinson, in his paper, dealt with the difficulty of the external cooling of gas engines and advocated the injection of water inside the cylinder. He contended that difficulties arising from corrosion and associated with lubrication could be overcome by regulating the quantity of water injected so that the temperature of the engine was maintained above 100 degs. C. Under these conditions the whole of the water injected was raised to boiling point on contact with the walls of the cylinder, and thus no accumulation of liquid was possible, and the drops of water would dissolve very little gas, such as sulphur dioxide, on their way to the walls because of their small surface and the short time—a mere fraction of a second—for which they were in contact with it, and what little they did absorb was expelled by evaporation. Experiments carried out by Dr. Dugald Clerk, the author, and others, had established the fact that the rate of heat-flow from the gas into the metal was more rapid at and soon after the moment of ignition than subsequently, and this led to the supposition that the heat flow into the bore of the cylinder during the last three-fourths of the stroke might be so small compared with that of the initial period that direct cooling of the last three-fourths of the cylinder could be omitted. And this supposition had been found to be correct, and the injection of water on to the surface of the combustion chamber and the piston effected the cooling of the whole of the barrel by conduction into the piston which was kept cool by projecting water onto the head when near the incentre. The author then described experiments with a Crossley engine with a cylinder 11½ in. in diameter, having a 21-in. stroke, rated at 40 b.h.p. at 180 r.p.m., to which a new cylinder without any water-jacket was fitted. The method had also been applied to larger engines, one of which was 18½-in. bore, rated at 105 b.h.p., and another of 36-in. bore, rated at 1000 h.p. All these trials were quite satisfactory and generally resulted in the attainment of a higher efficiency.

The discussion was opened by Engineer Rear Admiral E. Little, who was of opinion that if the author's method proved successful there was hardly any limit to the size of cylinder to which it could be applied. However, all the cylinders experimented with were of the horizontal type, and he wished to know if the principle could be applied to vertical engines.

Mr. M. Longridge said that the author's research was based on a scientific foundation, and although his method of cooling applied mostly to cylinders of comparatively large size it should be of value in smaller engines and make them cheaper and more economical.

Another speaker raised the question of erosion in the cylinder, consequent on deposits resulting from the evaporation of water.

Professor Hopkinson replied that he had not yet tried his method on vertical engines, but in the horizontal types with which he had experimented the solids deposited passed away by the exhaust valve. He wished to point out, too, that this method of cooling prevented pre-ignition.

Measurements of Temperature

Mr. R. S. Whipple's paper reviewed the various thermometric methods and instruments used mainly for technical purposes. He said that for steam plant mercury thermometers might generally be employed; but the thermometer in front of the smoke-stack and those on either side of the economizer were troublesome to read and were often neglected, and resistance thermometers which could be read on a galvanometer in the engine room would be preferable. Similarly, the resistance thermometer would be advantageous for cold storage work in which the opening of doors of the rooms for the purpose of reading mercury thermometers involved difficulty of maintaining a uniform temperature and entailed considerable cost for labor. With regard to pyrometers, one of their most important applications to the treatment of metals was the measurement of the temperature of the air in the hot-air mains of blast-furnaces; and for this purpose resistance thermometers with Callender recorders were best. To ensure an equal degree of accuracy with thermo-couples the scale control-board should be employed and due precaution adopted with regard to the cold junctions.

After dealing with the great difficulty of ascertaining the temperature of molten metal for casting, the author said that the heat treatment of metals afforded the greatest field for the use of pyrometers, and success depended on the way in which the pyrometer was mounted in the furnace. Experience had proved that with large furnaces it was advisable to place the pyrometer in the floor of the furnace, as advocated by Mr. F. Culpan.

With regard to the use of Seger cones in porcelain works, the author remarked that while they were of great utility in showing the maximum temperature reached, yet they afforded no index of the rate of heating, which was a most important matter.

For the lower temperature ranges thermo-couples were best, and for the higher ranges radiation pyrometers. The paper concluded by dealing at some length with the standardization of thermometers and pyrometers.

In the discussion Mr. C. Wicksteed remarked on the satisfactory results given by the apparatus described for determining temperatures in annealing furnaces. Dr. Glazebrook said that the greatest possible care was taken in making these instruments, and a very high degree of accuracy had been attained; but there were still difficulties at very high temperatures, and the problem was to find the necessary material for the production of an instrument for measurement up to 2000 degs. C. Sir Robert Hadfield said that at some Sheffield steel works about 5000 pyrometric readings were taken every week. The importance of temperature reading in steel works' practice was demonstrated by the great effects of comparatively small variations of temperature. It was necessary that an efficient method of ascertaining the temperature of molten metal should be available.

The British Association

The autumn meeting at Birmingham will open on the 10th September and continue until the 16th under the presidency of Sir Oliver Lodge. In the chemical section, Professor Soddy will discuss radio-active elements and the periodic law, while Dr. Desch, Dr. Holt, Professor Cohen, Professor Turner and Dr. Rosenhain, among others, will present papers of metallurgical interest.

In the physical science section Professor Barkla will contribute a paper on the nature of the X-rays.

The Institute of Metals

The summer meeting of the Institute of Metals will be held at Ghent on the 28th and 29th August. Professor A. K. Huntington will preside, and the second report to the corrosion committee by Dr. G. D. Bengough and Mr. R. Jones will be presented.

The International Electrotechnical Commission

The Berlin meeting of the Commission will extend from the 1st to the 5th of September. The first two days are to be devoted to special committees which will deal with the subjects of nomenclature, symbols, rating and prime movers. The official opening of the meeting will be on the 3rd, when the president, Dr. Budde, will deliver his address.

The Iron and Steel Institute

The autumn meeting will be held at Brussels, where the Grand Hall of the Palais des Académies has been placed at the disposal of the Institute for reading and discussion of papers on the 1st and 2nd of September. On the 3rd there will be a special excursion to the Ghent Exhibition; for the 4th, visits to the works of the Société John Cockerill, at Seraing; other works near Liège, to Charleroi and to Mons have been arranged.

Engineering Imports and Exports

The returns issued by the Board of Trade for the six months ended on the 30th of June show very satisfactory increases in both imports and exports of all the chief classes of engineering materials and products as compared with the corresponding period of 1912. Imports of iron and steel, including manufactures, amounted to £7,662,575, an increase of £1,948,711; and exports touched £227,932,692, an increase of £6,382,041. Imports of other metals, including manufactures, totalled £16,484,060, a rise of £1,967,937; and exports are valued at £6,898,155, an improvement of £1,355,117. In electrical goods the imports are put at £746,129, an increase of £24,308; and exports went up to £2,682,811, an increase of £616,478. Imports of machinery amounted to £3,757,738, a rise of £330,363, while exports to the value of £18,424,543 show an improvement of £2,735,357. New ships were imported to the amount of £14,345, an increase of £43; and were exported to the value of £4,323,731, an increase of £1,627,141.

The figures for the month of June alone are, excepting a small decline in electrical exports, of the same satisfactory character. Iron and steel values increased £291,164 in imports and £1,117,869 in exports; imports and exports of other metals rose £112,669 and £193,441, respectively; imports of electrical goods advanced £30,399, but exports declined £20,125; machinery showed increases of £51,393 for imports and £674,386 for exports; imports of new ships fell £632, but exports improved by £575,540.

Market Prices, July, 1913

	£ s. d.
Aluminium, ton	88. 0. 0
Alum, lump, loose, ton	6. 0. 0
Antimony, black sulphide powder, ton	20. 0. 0
Borax, British refined, crystal, ton	17.10. 0
Copper sulphate, ton	21.10. 0
Carbolic acid, liquid 97-99%, gal	1. 3
Caustic soda, ash 48%, ordinary, ton	5.10. 0
Ebonite, rod, per lb	4. 6
Hydrochloric acid, cwt	5. 0
India rubber, Para fine, lb	3. 8½
Mica, in orig. cases, medium, lb	3/6 to 6/–
Petroleum, Russian spot, gal	9¼
Quicksilver, bottle (Spanish)	7. 5. 0
Sal ammoniac, lump, 1sts, del. U. K., ton	44. 0. 0
Sulphate of ammonia, f.o.b. Liverpool, ton	12.16. 3
Sulphur, recovered, ton	5. 5. 0
Shellac, cwt	4.13. 0
Platinum, oz., nominal	9. 5. 0
Tin ore, ton	£115 to £117
Zinc, Vieille Montagne, f.o.b. Antwerp	24.15.0

Tin in the early part of the month dropped off considerably. Opening at 193, it had fallen by the 10th to £177; recovered till the 11th, £183; dropped again to £179.10.0 on the 15th and reached £185 on the 24th. Closes £182.

Copper remained about £63 till the 10th, when it dropped to £61.5.0 and remained flat till the 15th, when a steady rise set in. It reached 65½ by the 24th and continued very active, closing at £67.2.6.

Hematite remained flat at 72/4 for three days, then dropped to 71/6, and remained (with a slight dip on the 14th) at or about this level until the end of the month, closing 71/9.

Scotch Pig opened 65/– and closes lower at 64/-.

Cleveland opened 55/4, showed slight falling off from the 17th to 21st, but recovered 22nd. It closes flat at 54/10½.

Lead has risen steadily, opening at £19.15.0, reaching £20 by the 15th and closing £21.5.0.

DIFFERENCES

The principal feature in the markets for the above is the rapid rise in the price of copper, and the steady upward tendency of the price of lead.

	Higher. £ s. d.		*Lower.* £ s. d.
Ebonite rod	6	Aluminium	2. 0. 0
Quicksilver	5. 0	Borax, ton	1. 0. 0
Shellac	4. 0	Zinc	1. 2. 6
Copper	2. 17. 6	Tin	12. 0. 0
Lead	1. 10. 0	Hematite	1. 3
		Scotch Pig	1. 0
		Cleveland	7

The Great Falls Electrolytic Copper Refinery*

By Willis T. Burns

These notes are submitted, not as a discussion of the modern practice of electrolytic-copper refining, but as the record of a refinery that was among the pioneers in the field and that is to-day, and has been for 17 years, operating under conditions not to be found in any similar plant in the country. I refer to the current density, which is at least 60 per cent higher than that regularly employed elsewhere, and to the fact that the anodes treated are the direct product of the converter rather than of the reverberatory furnace.

This refinery, now one of the oldest plants of its kind operating in this country, was built in 1892 by the Boston & Montana Consolidated Copper & Silver Mining Company to refine the product of its Great Falls reduction works. The estimated capacity of the plant as designed was 65,000 lb. of cathodes per day at a current density of 16 amp per square foot.

The plant produced its first cathodes in February, 1893, and has been in almost continuous operation since that date. The output of the plant for the year 1893 was 3179 tons. As the smelter output increased, demands for a larger refinery output were made, and as no appropriation for enlarging the plant was forthcoming increased production could be obtained only by increasing the current density. As an abundance of water power was available and preliminary experiments had demonstrated that cathodes of satisfactory quality could be economically produced at more than double the density the plant was then operating at, new and larger generators were installed in 1896 and the current density brought up to 40 amp per square foot. Since that date the current density has been reduced by adding 76 refining tanks and by increasing the number of electrodes per tank.

In 1893, 3179 tons of copper were refined at an average current density of 10 amp per square foot; the production for the year 1912 was 31,506 tons and the average current density 34.1 amp per square foot.

General Arrangement of Refinery

As originally designed and constructed the refinery consisted of the tank house, a brick building 171 ft. 6 in. x 108 ft. (Fig. 1) containing 288 electrolytic-refining tanks of the following inside dimensions: 9 ft. 7 in. long, 2 ft. 4 in. wide and 3 ft. 9 in. deep, built of Western pine. The sides and bottoms were of 3-in. lumber and the ends of 2-in. material. The tanks were tied at the ends with 3-in. x ½-in. iron straps held together by 1-in. rods as shown in Fig. 2. The tanks were lined with ⅛-in. chemical sheet lead weighing about 8 lb. per square foot.

The tanks, which were supported in place by two 8-in. x 8-in. timbers which rested on stone piers 18 in. high, were insulated from the timbers by 4-in. x 4-in. x 1-in. glass insulators, eight under each tank. Had 5 ft. or 6 ft. of head room, instead of 18 in., been provided under the tanks, subsequent operations would have been greatly facilitated.

The tanks were arranged in 18 double rows, eight tanks deep (Fig. 1) with an aisle between each double row. The tanks as originally built were all on the same level, making it necessary to establish an individual system of circulation of the electrolyte for each tank. This was accomplished by providing two circulating tanks for each section of 96 tanks; these tanks were 10 ft. square by 8 ft. deep, lined with 12-lb. chemical sheet lead. One of these tanks was placed above the level of the refining tanks and one below, as shown in sketch. From the overhead tank the electrolyte was delivered, through an overhead system of lead pipes, to each separate refining tank. The solution was discharged into the refining tank at one end, near the bottom of the tank, and was allowed to overflow through a 1-in. lead pipe entering the tank 3 in. from the top near the other end. This pipe discharged into an open launder leading to the lower circulating tank, from where it was raised to the overhead tanks by various devices to be described later. This system, which permitted only a periodical circulation of the solution, soon gave way to a more efficient system to be described further on. Heating the electrolyte was effected by passing live steam through 125 ft. of 1-in. 8-lb. lead pipe in each of the overhead circulating tanks.

A ¾-in. x 4-in. steel trolley track was suspended from the roof timbers over each row of eight tanks. The anodes and cathodes were removed in and out of the tanks by means of chain blocks suspended from two-wheel trolleys on this track.

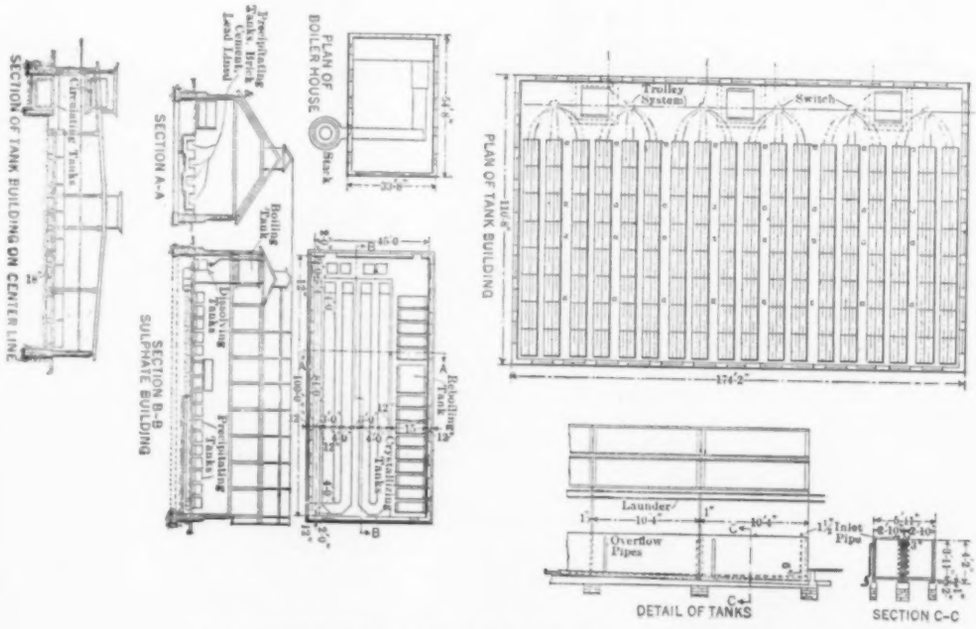


FIG. 1—GREAT FALLS ELECTROLYTIC PLANT AS ORIGINALLY BUILT

The current was distributed by means of ¾-in. x 4-in. rolled-copper busbars attached to the sides of the tanks with wooden brackets. The tanks were all connected in series and the electrodes in multiple.

The original installation included a sulphate plant for the manufacture of bluestone and a boiler house containing one 100-hp locomotive-type boiler to furnish steam for heating and boiling solutions.

The sulphate plant, shown in Fig. 1, was well appointed for the manufacture of bluestone as described later. The capacity of this plant was 2,500,000 lb. of bluestone per year.

The above is a brief description of the plant as it originally existed. Fig. 3 is a sketch of the plant as it exists to-day. One division of 32 refining tanks has been added to the main tank room, 44 new electrolytic refining tanks for the production of starting sheets have been housed in additions as have the 10 new insoluble-anode tanks shown in sketch. An addition has been built south of the tank room in which is installed the equipment for pumping and heating the electrolyte. An addition has been built on the east containing the scale house

*Extracts from a paper read before the American Institute of Mining Engineers in Great Falls, Mont., on August 16th. The complete paper is published in the August issue of the American Institute of Mining Engineers, pages 2011 to 2049. As given above, the paper is practically complete in all main points, with exceptions of the chapters on the transmission line and on physical testing.

and a 7-ton traveling crane for handling the cathodes and scrap after weighing. The addition on the north side of the building contains store room, shops and office. Fig. 4 is a flow sheet of the electrolytic plant.

In the new building south of the bluestone building are the carpenter and lead-burning shops and the machinery used in trimming sheets. The four large round tanks situated further south are for the storage of solutions.

The original boiler house has been replaced by a brick building containing two 150-hp Heine and one 100-hp Scotch marine

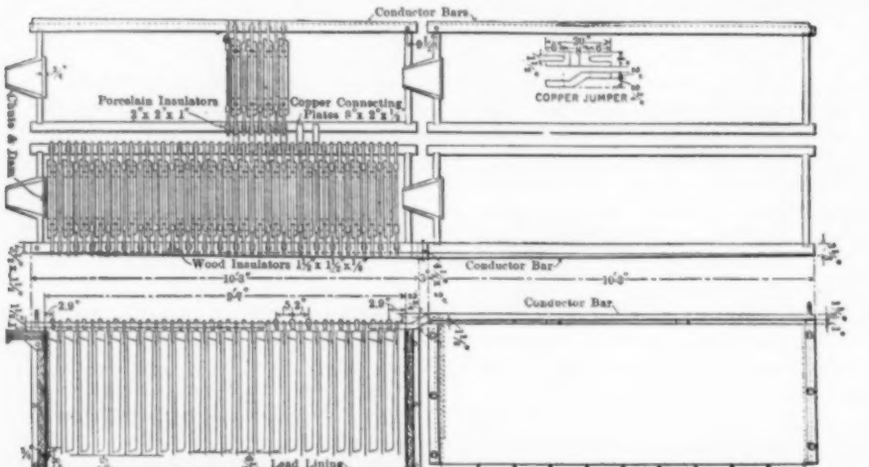

Conductor Bars
Porcelain Insulators 2" x 2" x 1"
Copper Connecting Plates 6" x 2" x ⅛"
COPPER JUMPER
Chute & Dam
Wood Insulators 1½" x 1½" x ¼"
Conductor Bar
Conductor Bar
Lead Lining
8" x 8" Glass Insulators 4" x 4" x 1"
Concrete Pier

FIG 2—TANK DETAILS

boiler from which steam for heating and boiling solutions is obtained.

As previously stated the refining tanks, as originally installed, were all on the same level and to obtain a circulation of the electrolyte it was necessary to deliver the electrolyte to each tank through a separate pipe line. This system was found unsatisfactory and the refining tanks were rearranged in a cascade of eight tanks with a drop of 2¾ in. between tanks as shown in Fig. 3. The electrolyte is now delivered to the upper tank of the cascade at the surface and is drawn off from a point 8 in. from the bottom at the lower end of the tank by means of the dam shown and is delivered to the next tank through the cast-lead antimony chute shown in Fig. 2. From the last or lower tank of the cascade the electrolyte is returned to the pump room through a 2-in. 7-lb. lead pipe attached to the sides of the tanks.

The original refining tanks were lined with ⅛-in., 8-lb. chemical sheet lead and the overflow consisted of a 1-in. lead pipe burned to the lining. Tanks constructed in this manner were found to last about 10 years in service. As they gave out they were replaced by tanks of the same dimensions lined with ⅛-in., 7-lb. lead containing 6 per cent antimony. The cast chutes of the same material were built into the ends of the tanks. Tanks thus constructed have now been in service for 11 years and do not show much deterioration. The antimonial lead is harder and has a less tendency to creep and buckle under extreme changes of temperature than is the case with the chemical lead. Practically all of our tanks are now lined with antimonial lead.

When the change from 4000 amp to 9000 amp was made in 1896 it became necessary to increase the size of the tank busbars. The ¾-in. x 4-in. rolled-copper conductors were removed from the sides of the tanks and a taper bar 4½ in. x 2 in. in the center and 1.5 in. x 3 in. at the ends was laid on the top of the tanks, as shown in Fig. 2. The bars were cast at our works from wire-bar copper. These bars are insulated from the tanks by means of ½-in. x 4-in. pine strips previously boiled in pine tar. These strips are renewed once in six months.

Into the ends of the busbars ⅞-in. x 6-in. iron studs are screwed to receive the copper shunt used to cut four tanks out of circuit for removal of slime.

The anode and cathode rods and the connection plates, used between the tanks, are also cast at our furnace refinery in copper molds. Details of tank connections are shown in Fig. 2.

Anodes

As previously mentioned this refinery is the only plant of its kind treating converter-copper anodes. The general practice elsewhere is to transfer the molten copper from the converter to the reverberatory furnace. Here the necessary oxidizing and poling of the copper to bring it to the proper pitch for casting takes place. The copper is then cast into anodes by means of a suitable casting machine.

This refining process has the effect of increasing the copper contents of the metal by from 0.3 per cent to 0.4 per cent, sulphur dioxide being the principal impurity eliminated. The resulting anode is much denser than the converter anode and the casting is free from the uneven surfaces which appear on the converter anode. Analyses of anodes are shown in Fig. 5.

Table I shows one of a number of comparisons, made at this plant, between converter and refined anodes. In conducting this test three divisions, of 32 refining tanks each, were employed. One-half of the tanks in each division were treating refined anodes while the other half contained converter anodes. By this arrangement the effect of the personal equation of the men in charge of the tanks was reduced to a minimum.

It will be noted that while the ampere efficiency of deposit was 3.6 per cent higher in the refined-anode tanks than in the

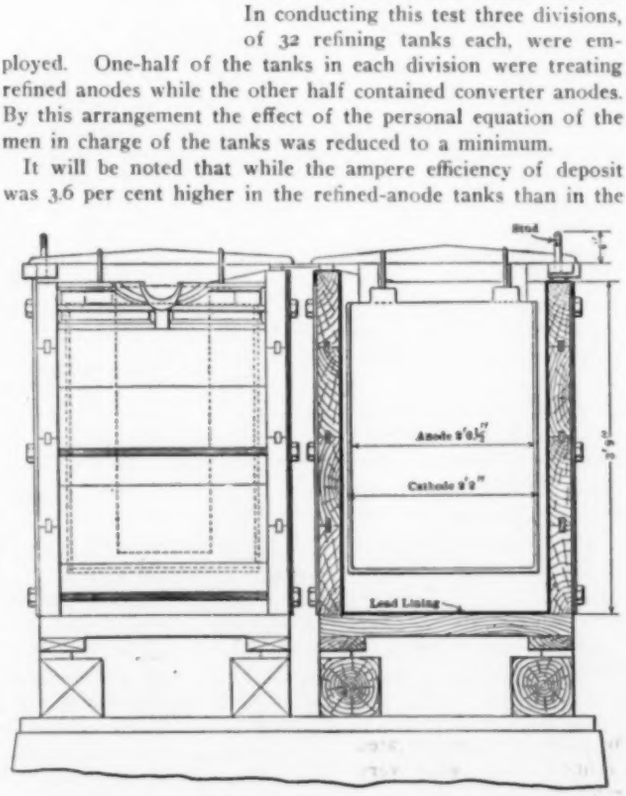


FIG. 2A—TANK DETAILS

tanks containing converter anodes the production per kilowatt-hour was slightly higher in the converter-anode tanks. Subsequent experiments have shown that the refined anodes admit of closer spacing than do converter anodes, without the same reduction in ampere efficiency, the effect of which is to decrease the resistance and increase the production per kilowatt. The difference, however, was not found to be great.

The advantages in favor of the refined anodes, as they appear

in Table I, are: First, higher grade of slime resulting in a saving in cost of slime refining; second, lower silver contents of cathodes, all of the silver in the cathode representing a dead loss; third, the lower percentage of anode scrap.

When reduced to dollars per ton of copper refined the sum of the savings that would result from the use of refined anodes in a plant of this type was found to be less than one-half the cost per ton of the reverberatory-furnace treatment of the copper. The cathodes produced from the two types of anodes differed but little in physical appearance.

TABLE I.—COMPARISON OF CONVERTER AND REFINED ANODES CAST IN THE SAME MOLDS

	Converter Anodes	Refined Anodes
Number of days covered by test	50	50
Number of refining tanks employed	48	48
Average analyses of anodes:		
Per cent. Cu	98.91	99.27
Per cent. As + Sb	0.072	0.071
Oz. Ag per ton	59.09	61.14
Oz. Au per ton	0.200	0.219
Average analyses of electrolyte:		
Specific gravity	1.20	1.20
Grams per liter Cu	43.5	43.5
Grams per liter free acid	160	160
Grams per liter As	11.97	11.97
Grams per liter Sb	0.49	0.49
Grams per liter Fe	10.09	10.09
Grams per liter Cl	0.045	0.045
Average temperature of electrolyte:		
Inlet of 8-tank cascade, C°	58	58
Outlet of 8-tank cascade, C°	54	54
Rate of circulation of electrolyte, gal. per min.	6	6
Number of anodes per tank	20	20
Number of cathodes per tank	20	20
Average weight per new anode, lb.	525	632
Average thickness per new anode, in.	3	3
Distance, center of anode to center of cathode, in.	2.87	2.87
Active cathode surface per tank, sq. ft.	252	252
Average amperes per tank	8387	8387
Average volts for 48 tanks	27.21	28.53
Average volts per tank	0.567	0.594
Average kilowatts for 48 tanks	228.2	239.3
Total copper deposited, lb.	1,103,749	1,148,749
Average age of cathodes drawn	2½	2½
Average ampere efficiency of deposit, per cent.	88.3	91.9
Average amperes per sq. ft. cathode surface	33.3	33.3
Average lb. copper deposited per kilowatt-hour	4.03	4.00
Average oz. per ton silver in cathodes	1.25	.95
Average per cent. As + Sb in cathodes	0.0043	0.0043
Average per cent. anode scrap	8.00	5.30
Analyses of silver slime:		
Per cent. Cu	40.3	18.80
Oz. Ag per ton	6755.00	14079
Oz. Au per ton	18.34	38.45

scrap resulting from each anode weighs about 30 lb.; this is resmelted in the converters. The clip type of anode also has the advantage, as compared with the lug type, of being suspended in such a manner as always to hang plumb in the tank. The standard converter anode measures 24.5 in. by 35.75 in.

The clip or hanger is made from a ¼-in. round copper rod 60 ft. long rolled in a rod mill from a bar cast at the refining furnace. This rod is cut and bent into the loop form, in an automatic machine, and placed in the anode mold. Both the anodes and cathodes are supported in the tanks by cast-copper rods as shown in sketch. The anodes are spaced 5.2 in. center to center. It will be noted in Fig. 6 that the anode is 3 in. thick at the top and 2.5 in. thick at the bottom. It is found that the anode corrodes more rapidly at the upper end where the current enters than at the lower end, that the wedge shape yields a lower percentage of scrap and that the anode retains its original shape for a longer time than is the case with an anode of uniform thickness.

Cathodes

The starting sheets are produced in 44 standard refining tanks divided into two circuits connected in parallel in the main circuit and are thus operating at one-half the current density of the commercial refining tanks. Each tank contains 21 cathode blanks and 22 anodes. The blanks are made of ¼-in. rolled copper 27.5 in. x 39.5 in. below the solution line, as shown in Fig. 6.

The anodes used in these tanks are of the same type as the standard anode but larger, being 26.5 in. x 38⅝ in.

The 12-hour sheets are stripped at an average weight of 4 lb. each, measuring 26 in. x 36.5 in. after trimming. The men employed in stripping the plates do their work directly over the tanks, using a traveling bench suspended from the trolley track. This bench, on which the sheets are piled after stripping, is moved from tank to tank as the men advance. The blanks are covered with a thin coating of low-grade mineral oil to which 0.5 lb. of artificial graphite per gallon of oil is added. Grooved wooden strips are used on the edges of the plates to prevent deposition of copper. The strippers work independently and each man strips and delivers to the trimmer 360 sheets per eight-hour shift. The electrolyte used in these tanks differs slightly from that of the main tank room. An average analysis is given in Table II.

In a crane-operated electrolytic refinery, where the anodes and cathodes are handled in tank units and not individually as here, difficulties would result if converter anodes were used because of the fact that converter anodes corrode less uniformly than refined anodes. Many of the converter anodes would become scrap before the date set for drawing the scrap from a tank and an equal number of pieces would not be scrap at the time for drawing.

The above comparison, together with other tests made at this plant, has demonstrated that refined anodes, while very desirable, would for this plant be an expensive luxury.

Fig. 6 shows the earlier form of anode used in this plant and the Morrow clip type[1] now used. The advantage of the clip type of anode is that the amount of inactive copper above the solution line is much less than is the case with the lug type. The average percentage of anode scrap for the year 1912 was 5.9; the average weight of the converter anode is 500 lb. The

[1] U. S. patents No. 621,121 and 631,471.

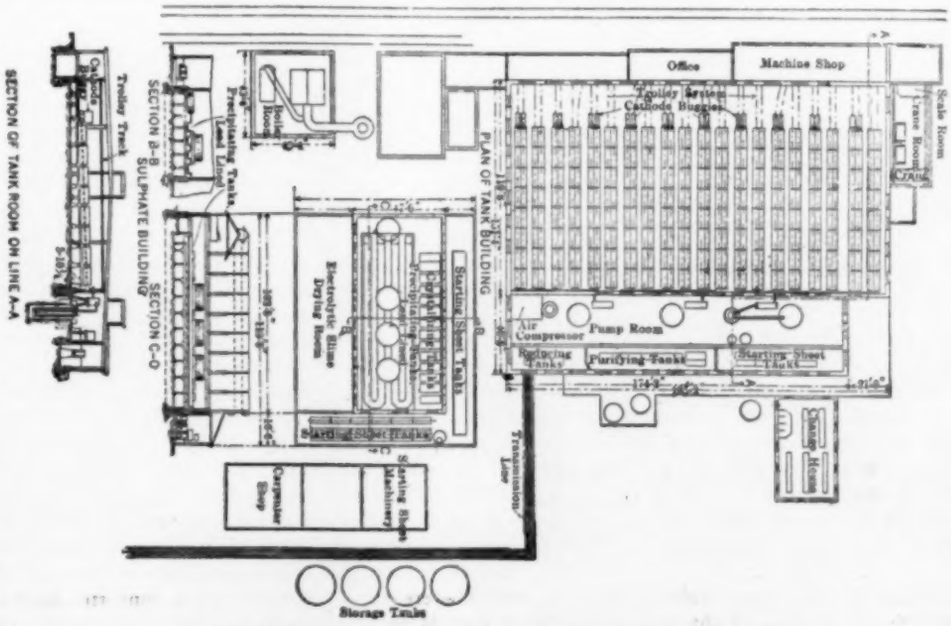


FIG. 3—GREAT FALLS ELECTROLYTIC PLANT AS REMODELED

TABLE II.—STARTING-SHEET TANK ELECTROLYTE

Specific gravity	1.175
Free H_2SO_4, grams per liter	120.0
Cu, grams per liter	40.0
As, grams per liter	5.0
Sb, grams per liter	0.4
Fe, grams per liter	4.5
Cl, grams per liter	0.04

Table III shows the effect of sulphuric acid in the electrolyte on the efficiency of deposition, the production per kilowatt-hour increasing as the acid increases. The sheets produced when the acid is about 120 g. per liter are found to be smoother and tougher than when acid is higher.

The starting sheets are trimmed on the four edges to prevent short circuits. Two clips or loops, cut from similar sheets, are then attached to each sheet by means of the Morrow clip machine[2] (see Fig. 6).

TABLE III.—EFFECT OF FREE SULPHURIC ACID ON STARTING-SHEET PRODUCTION

Converter anodes, 39.5 in. by 27.5 in. by 0.25 in. blanks, 3 in. anode; 2.5 in center of anode to center of blank

Circulating electrolyte at 4 gal. per minute

Amperes	Amps. per Sq. Ft.	Efficiency	E.M.F. per Tank Volts	Lb. Cu. per Kilo-hour	Average Weight per Sheet	Electrolyte		
						Gal. per Liter Free H_2SO_4	Cu	Temperature C°
4,580	16.2	86.5	0.475	4.75	5.86	74	42.6	52.5
4,490	15.9	86.5	0.462	4.84	5.78	74	42.6	52
4,453	15.7	85.2	0.462	4.80	5.61	70	40.1	55
4,450	15.7	83.6	0.475	4.55	5.50	69	39.8	49
4,390	15.5	90.3	0.482	4.84	5.87	69	40.1	50
4,403	15.6	90.2	0.496	4.71	5.87	73	38.4	47.5
4,290	15.2	88.2	0.462	4.94	5.60	80	38.9	51
4,397	15.5	91.6	0.470	5.04	5.93	82	38.8	52
4,538	16.0	90.3	0.476	4.90	6.06	88	38.8	51
4,461	15.8	91.9	0.450	5.28	6.06	89	39.9	52
4,408	15.6	92.5	0.426	5.63	6.03	110	40.4	52
4,506	16.0	92.5	0.452	5.29	6.17	109	38.9	53
4,496	15.9	91.7	0.424	5.55	6.10	116	40.6	53
4,512	16.0	90.3	0.426	5.50	6.02	126	40.5	50
4,550	16.1	89.7	0.420	5.53	6.04	128	40.4	48
4,430	15.7	90.7	0.400	5.86	5.94	129	42.6	48
4,435	15.7	84.7	0.385	5.69	5.56	128	42.0	48
4,420	15.6	87.3	0.365	6.09	5.62	126	42.4	50
4,445	15.7	91.8	0.357	6.23	6.03	124	43.6	48
4,453	15.7	85.1	0.360	6.13	5.60	132	42.1	46
4,404	15.6	87.0	0.342	6.55	5.66	148	40.8	49
4,450	15.7	88.6	0.350	6.62	5.82	149	42.3	46

Note—The sheets produced with between 116 and 126 g. per liter of acid were the toughest and best.

The starting sheets are now ready for the tanks. It will be noted that the cathodes hang ¾ in. lower in the tank than the anode and are 1.5 in. wider than the anode. When using anodes and cathodes of the same size a cathode very rough on the edges resulted; increasing the size of the cathode has resulted in a cathode with comparatively smooth edges and an increase of 5 per cent in ampere efficiency.

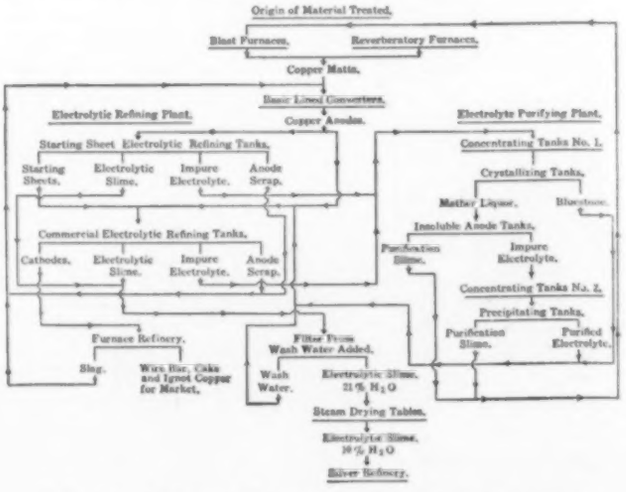


FIG. 4—FLOW SHEET OF GREAT FALLS ELECTROLYTIC PLANT

When the output of the generators is 2000 kw or more the best results are obtained while drawing two and three-day cathodes. With a lower generator output older cathodes are drawn. The generator output is governed by the head on the water wheels.

[2]U. S. patent 600,498, Mar. 8, 1898.

To determine the most economical age of cathodes and number of electrodes per tank at which to operate, the labor, the production per kw-hour, the silver lost in cathodes and the impurities in the anodes and cathodes must be considered. Assuming the quality of the product as regards impurities to

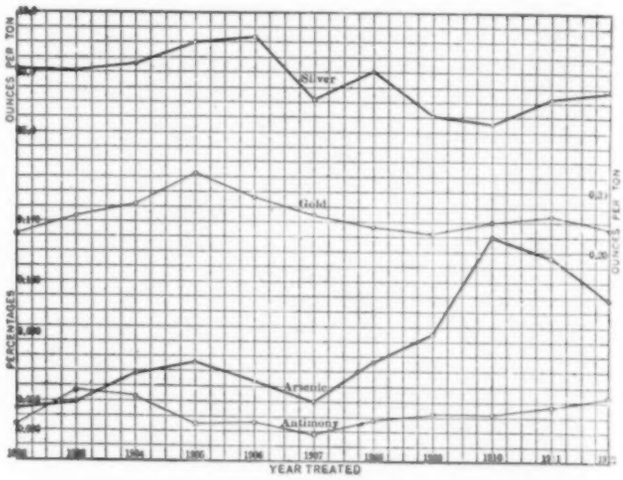


FIG. 5—CONVERTER ANODES

remain satisfactory under the different conditions the cost per ton remains as the only consideration. Table IV is a summary of various comparisons made between different ages of cathodes with varying numbers of electrodes per tank.

TABLE IV.—EFFECT OF VARYING THE AGE OF CATHODES AND THE NUMBER OF ELECTRODES PER TANK

Age of Cathodes, Days	Electrodes per Tank		Average Amperes	Average Amperes per Sq. Ft.	Ampere Efficiency, Per Cent.	Cu per Kilowatt-hour, Lb.	Cathode Oz. Ag per Ton	Analyses As + Sb, Per Cent.
	Anodes[a]	Cathodes						
4	20	20	9,300	36.9	88.0	3.93	1.32	0.0036
2	20	20	8,808	35.0	90.85	3.72	0.83	0.0030
3	20	20	8,877	35.2	89.00	3.75	0.83	0.0032
2	21	21	9,035	34.1	90.90	3.84	0.89	0.0032
3	21	21	9,223	34.8	89.40	3.87	1.02	0.0029
2	22	22	9,071	32.6	90.50	4.02	0.89	0.0033
3	22	22	9,167	33.0	88.80	4.07	0.95	0.0030

[a]Converter anodes. Average analysis of anodes: Cu, 99.13; As, 0.127; Sb, 0.055 per cent.; Ag, 33.97 oz. per ton; Au, 0.22 oz.

Table V is the comparison, in detail, between 21 and 22 pairs of electrodes per tank while drawing two and three-day cathodes.

The Electrolyte

The 320 commercial refining tanks are divided into three sections, two sections containing 96 tanks each and the third section 128 tanks.

Each section is provided with a separate electrolyte and circulating system. Copper and acid determinations are made daily on samples of each electrolyte. As, Sb, Fe and Cl determinations are made weekly.

The two groups of starting-sheet tanks each have a separate electrolyte which is sampled and assayed in the same manner as the commercial tanks electrolyte.

Some analyses of the electrolyte in the commercial refining tanks are shown in Fig. 5. The analysis of solutions is ordinarily reported in terms of grams per liter. With a current density of 34 amp per square foot and a circulation of 6 gal. of electrolyte per minute the best results, while treating converter anodes, are obtained with an electrolyte carrying 40 g. per liter of copper and 160 g. per liter of free sulphuric acid.

The circulation of the electrolyte is through a cascade of eight tanks as already described. In the pump room the electrolyte is delivered to the receiving tank, a round lead-lined tank 11 ft. in diameter and 4.5 ft. deep (see Fig. 3). In this

TABLE V.—EFFECT OF VARYING THE AGE OF CATHODES AND NUMBER OF ELECTRODES PER TANK

Details of 21 and 22 pairs of electrodes per tank in Table V. Standard Converter Anodes

	21 Anodes and 21 Cathodes per Tank	22 Anodes and 22 Cathodes per Tank
Number of refining tanks used	160	160
Copper deposited, 2-day cathodes, lb	1,778,352	1,704,743
Copper deposited, 3-day cathodes, lb	1,742,678	1,889,401
Total copper deposited, lb	3,521,030	3,594,144
Spacing of electrodes, anode centers, in	5.45	5.20
Average amperes for 2-day cathodes	9,035	9,071
Average amperes for 3-day cathodes	9,223	9,167
Average amperes per sq. ft., 2-day cathodes	34.1	32.6
Average amperes per sq. ft., 3-day cathodes	34.8	33.0
Average ampere efficiency, 2-day cathodes, per cent	90.9	90.5
Average ampere efficiency, 3-day cathodes, per cent	89.4	88.8
Average drop per tank in volts, 2-day cathodes	0.612	0.581
Average drop per tank in volts, 3-day cathodes	0.600	0.562
Average copper deposited per kilowatt-hour, 2-day cathodes, lb	3.837	4.021
Average copper deposited per kilowatt-hour, 3-day cathodes, lb	3.871	4.075
Average copper deposited per kilowatt-hour, 2- and 3-day cathodes, lb	3.854	4.048
Analyses of cathodes:		
2-day cathodes, As + Sb, per cent	0.0032	0.0033
3-day cathodes, As + Sb, per cent	0.0029	0.0030
2-day cathodes, Ag, oz. per ton	0.89	0.89
3-day cathodes, Ag, oz. per ton	1.02	0.95
	Same both set of Tanks	
Analyses of Electrolyte:		
Specific gravity	1.219	
Cu, grams per liter	41.1	
Free acid, grams per liter	153	
As, grams per liter	6.49	
Sb, grams per liter	0.51	
Fe, grams per liter	5.11	
Cl, grams per liter	0.44	
Average temperature of electrolyte (8 tanks in cascade):		
Inlet tank, C°	57	
Outlet tank, C°	53	
Speed of circulation of electrolyte, gal. per min.	6	
Analyses of anodes (converter):		
Cu, per cent	99.13	
As, per cent	0.127	
Sb, per cent	0.055	
Ag, oz. per ton	33.97	
Au, oz. per ton	0.22	

tank are 150 ft. of 1-in. 8-lb. antimonial lead pipe; live steam is supplied to these coils at a pressure of 65 lb. per square inch, by means of which the electrolyte is heated. From the receiving tank the electrolyte is raised to the discharge tank from which it is again conducted to the head tank of the cascade.

Various means of elevating the electrolyte have been employed in this plant. The first method used was the steam injector constructed of lead and antimony. These pumps were of limited capacity and the dilution of the solution was excessive. They were followed by a system of lead-lined cast-iron eggs in which the electrolyte was collected and blown to the overhead tank with compressed air. This system was very inefficient and gave way to a type of boiler-feed pump, the water end of which was of bronze. These soon became leaky from corrosion and were abandoned. A type of plunger pump with a lead-lined barrel and rubber valve and plunger was next tried. This pump was continued in service for several years with satisfactory results but was replaced 10 years ago by the air lift, which has continued to hold the field. The air lift as used here is constructed of cast lead-antimony pipe 6 in. inside diameter, ¾-in. walls. The pipe is cast in 4-ft. lengths with flanged joints. A brick wall 20 ft. deep and 4 ft. in diameter, made waterproof, with a cement lining, contains that portion of the lift that is below the floor level. Working against a head of 14 ft. 8 in., 160 gal. of electrolyte (1.22 specific gravity) is raised per minute with a consumption of 80 cu. ft. of free air compressed to 16 lb. per square inch (see Fig. 7).

The lift, while it is of low mechanical efficiency and sensitive to variations in the air pressure, is very satisfactory in its operation, having no moving parts and requiring but little attention. The loss in temperature of the electrolyte during its passage through the air lift from the receiving tank to the discharge tank averages 0.5° C.

TABLE VI.—EFFECT OF SPEED OF CIRCULATION OF ELECTROLYTE ON THE CONCENTRATION OF THE SOLUTION

Current density 36.0 amperes per square foot, 4-day cathodes, converter anodes. Circulating Electrolyte at Rate of 6 gal. per minute

Zone	Specific Gravity	Grams per Liter						Copper per Kilowatt-hour Lb.
		Acid	Cu	As	Sb	Fe	Cl	
1	1.213	174	38.4	7.5	0.56	2.95	0.044	.....
2	1.216	172	39.2	7.2	0.59	2.95	0.045	.....
3	1.216	171	40.2	7.1	0.59	2.95	0.043	.....
4	1.228	153	49.9	7.2	0.59	2.90	0.044	.....
Av'ge	1.218	168	41.9	7.3	0.58	2.94	0.044	3.92
Circulating Electrolyte at the Rate of 4 gal. per minute								
1	1.211	163	40.0	7.3	0.52	2.90	0.042	.....
2	1.214	161	41.1	7.3	0.55	2.92	0.040	.....
3	1.216	159	42.0	7.4	0.58	2.95	0.040	.....
4	1.263	139	67.8	7.4	0.55	2.90	0.040	.....
Av'ge	1.226	156	47.7	7.4	0.55	2.92	0.040	3.85
Circulating Electrolyte at rate of 2 gal. per minute								
1	1.210	167	37.8	7.6	0.52	2.90	0.042	.....
2	1.213	165	39.0	7.1	0.49	2.95	0.042	.....
3	1.217	162	41.0	7.6	0.51	2.95	0.042	.....
4	1.265	146	66.4	7.9	0.55	2.95	0.041	.....
Av'ge	1.226	160	46.1	7.6	0.52	2.94	0.042	.....
Sampled after circulation had been entirely shut off for 7 hr. with current passing at the rate of 36 amperes per square foot								
1	1.185	179	23.4					
2	1.210	164	38.0					
3	1.230	151	51.8					
4	1.255	138	65.5					
Av'ge	1.220	158	44.7					

The compressed air for the operation of the lift is obtained from the converter-plant blowing engines which are operated by water power. A motor-driven compressor at the tank house supplies air in cases of emergency.

Circulation of the electolyte is maintained at a speed of 6 gal. per minute, a lower circulation reducing the production per kw-hour while a higher rate has the effect of disturbing too much the settlement of the slime and increasing the silver in cathodes. Table VI shows the effect on the concentration of the electrolyte of varying the speed of circulation. The samples which were taken in the regular refining tank and marked zones 1, 2, 3 and 4, were obtained in the following manner: A glass tube, closed at the top, was lowered to a point 6 in. below the surface of the electrolyte and a sample collected at this point and called zone 1. Zone 2 sample was taken in the same manner at a point 17 in. below the surface; No. 3, 28 in. below the surface and No. 4, 39 in. below the surface. The column of electrolyte in the tanks was 40 in. high, there being 3 in. of slime in the tanks. These figures represent the averages of 104 samples taken during seven consecutive days from 48 refining tanks at 36 amp per square foot. The samples were all taken between the center pair of electrodes.

These figures show a rather remarkable concentration of the electrolyte as regards copper and acid with negative results regarding impurities in solution and demonstrated that, while operating at 36 amp per square foot, circulating the electrolyte at the rate of 6 gal. per minute delivering the solution at the surface of the tank and discharging from a point near the bottom, fails to maintain the electrolyte in a homogeneous state but is found to be the most economical speed at which to operate.

The arsenic and antimony are the impurities with which we are chiefly concerned in our refining operations, and as their effects on the conductivity of the copper are much alike our practice is to report them combined as arsenic plus antimony in the cathodes, making a separation on the average monthly sample to note the relations. These impurities in the electrolyte and in the anodes are determined separately.

There is no silver in solution in the electrolyte and all of the

silver present in the cathodes is deposited mechanically from the slime in suspension; the slime is entrapped on the rough surfaces of the cathodes, the silver contents increasing with the age and roughness of the cathode. The arsenic and antimony are deposited both mechanically from the slime and electrolytically from the electrolyte.

When refining converter anodes carrying 0.12 per cent As, at 34 amp per square foot the arsenic in the electrolyte is not allowed to exceed 7 g. per liter.

The results of an experiment with covered cathodes and converter anodes may be of interest. Three regular cathode starting sheets were placed in separate frames covered with 8-oz. duck in such a manner as to entirely inclose the sheets. These were substituted for three regular cathodes in a commercial refining tank and after remaining 2.75 days they were removed, sampled and assayed as shown in Table VII.

TABLE VII.—ANALYSIS OF ELECTROLYTE

Specific Gravity	Grams per Liter					
	Acid	Cu	As	Sb	Fe	Cl
1.200	152	41.2	7.0	0.44	1.80	0.04

ANALYSIS OF ANODES

Per Cent. Cu	Oz. Ag per Ton	Oz. Au per Ton	Per Cent. As	Per Cent. Sb
99.15	39.0	0.25	0.211	0.04

CATHODES PRODUCED

	Inclosed Cathode	Regular Cathode
Current density, amperes per sq. ft.	31.6	35.7
Age of cathodes, days	2.75	4.0
Ag, oz. per ton	0.2	1.2
As + Sb, per cent	0.0036	0.0042

The lower circuit density of the covered cathodes is due to the increased resistance offered by the covering.

The canvas-covered cathodes were drawn at 2.75 days rather than four days as the canvas was showing signs of failure. The results show that the canvas covering was effective in preventing but little of the silver-bearing slime in suspension from coming in contact with the cathodes but that it had a lesser effect on the arsenic and antimony contents of the deposited copper.

The original method employed at these works of purifying the electrolyte was the manufacture of bluestone. An amount of electrolyte sufficient to maintain the working solutions at the required degree of purity was daily run off from the tank house to the sulphate plant. Here it was heated to the boiling point by means of steam coils and passed through open tanks containing shot copper into which compressed air was blown until all of the free acid was neutralized. After the necessary concentration by boiling the solution was allowed to cool in an open chute where most of the copper crystallized out. The crystals from the chute were redissolved in water, concentrated, and run into the final crystallizing tanks. The crystals from these tanks were washed, dried and packed in barrels for shipment. The mother liquor, after it had become too foul for further use, was run over scrap iron for the recovery of the small amount of copper remaining in solution. After passing over the iron the solution, which contained only the impurities, was wasted.

This was an efficient method of freeing the electrolyte of impurities and would probably still be employed had not the market for the bluestone failed during 1899. In 1897 there were 2,300,000 lb. of bluestone produced and shipped from our sulphate plant.

The electrolytic method was next employed for purifying the electrolyte. This process consisted in removing from the working solutions each day such an amount of electrolyte as was necessary to maintain the purity of the electrolyte. This solution was passed through four standard refining tanks containing lead anodes and copper or lead cathodes. These tanks were arranged in a cascade the electrolyte flowing from one tank to another. The Cu, As and Sb contents of the solution, as it left the last tank, depended on the speed of flow. With a circulation of 2 liters per minute but a trace of Cu, As and Sb remained. The ampere efficiency of deposit, however, was extremely low at this rate of flow.

Table VIII shows the results obtained while circulating at 4 liters per minute. The percentages of elimination as shown

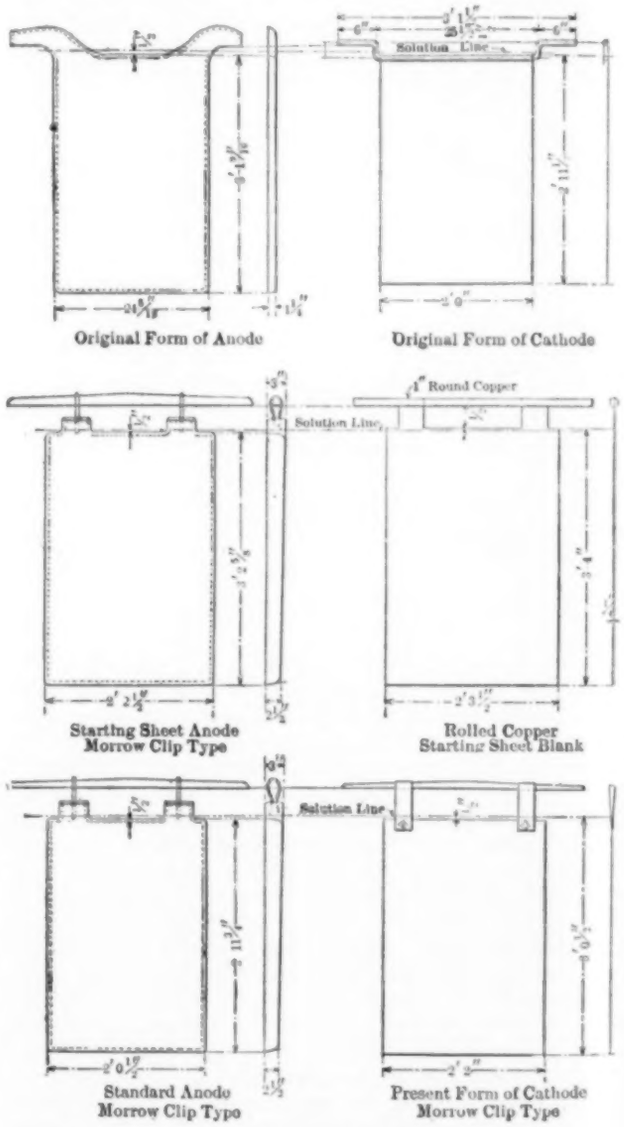

Solution Line
Original Form of Anode
Original Form of Cathode
1" Round Copper
Solution Line
Starting Sheet Anode Morrow Clip Type
Rolled Copper Starting Sheet Blank
Solution Line
Standard Anode Morrow Clip Type
Present Form of Cathode Morrow Clip Type

FIG. 6—ANODES, CATHODES, AND STARTING SHEET BLANK

in this table are high while the ampere efficiency is very low, the two lower tanks in the cascade doing but little work. In subsequent experiments the most economical speed of flow was found to be 7 liters per minute with an elimination of 99 per cent of the copper, 78 per cent of the arsenic and 91.1 per cent of the antimony with a total ampere efficiency of about 50 per cent.

The increase in the iron contents of the electrolyte between the inlet of the first and outlet of the last tank, as shown in Table VIII, is due to the concentration of the electrolyte by reason of the heat evolved by the electrolysis. These analyses are corrected on the basis of an unchanged volume of electrolyte in Table IX.

TABLE VIII.—REMOVAL OF COPPER, ARSENIC AND ANTIMONY FROM ELECTROLYTE IN INSOLUBLE ANODE TANKS

Circulation, four liters per minute. Lead anodes, copper cathodes, 9,000 amperes, 31.8 amperes per square foot

	Grams per Liter					Volts per Tank	Temperature, C°
	Acid	Cu	Fe	As	Sb		
Inlet tank No. 1	144	37.060	6.242	3.200	0.463	.....	17
Outlet tank No. 1	184	7.376	6.813	2.240	0.260	2.22	42
Outlet tank No. 2	194	0.504	7.364	0.400	0.061	2.25	57
Outlet tank No. 3	208	0.088	7.701	0.056	0.038	2.25	64
Outlet tank No. 4	216	0.048	7.915	0.028	0.028	2.25	65

TABLE IX.—CORRECTED ANALYSES

	Grams per Liter					Percentage Elimination of Original Amounts			Ampere Efficiency Per Cent.
	Acid	Cu	Fe	As	Sb	Cu	As	Sb	
Inlet tank No. 1	144	37.060	6.242	3.200	0.4630	.....	.....	.....	.....
Outlet tank No. 1	169	6.760	6.242	2.050	0.2380	81.8	35.9	48.7	71.70
Outlet tank No. 2	165	0.427	6.242	0.339	0.0517	17.1	53.5	40.2	19.50
Outlet tank No. 3	169	0.071	6.242	0.045	0.0308	0.9	9.2	4.7	1.68
Outlet tank No. 4	170	0.038	6.242	0.022	0.0220	0.1	0.7	1.7	0.15
Totals and averages	.....	.....	.....	.....	.....	99.9	99.3	95.3	23.26

The process thus far described has removed only the Cu, As and Sb from the solution, leaving the Fe, Ni, Bi and Zn. To remove these the electrolyte, as it leaves the insoluble-anode tanks, is run into a lead-lined tank 13 ft. in diameter and 4.5 ft deep lined with 12-lb. chemical sheet lead and containing 600 ft. of 1-in., 8-lb. chemical-lead pipe. In this tank it is concentrated to 55° Bé., decanted to an open tank 10 ft. long, 4 ft.

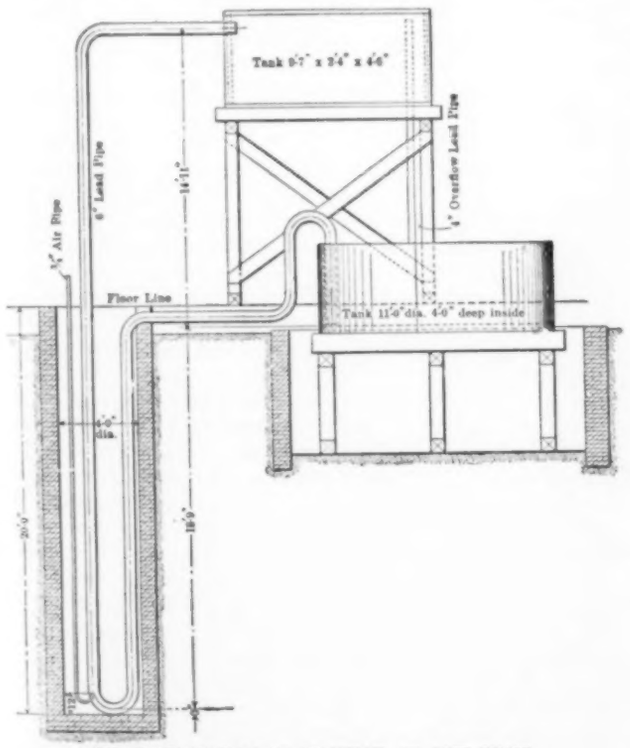


FIG. 7—ELEVATION OF SOLUTION AIR-LIFT PUMP

wide and 3 ft. deep, where it is allowed to stand for four days, during which time the salts of Fe, Ni, Bi and Zn crystallize out, leaving a solution of approximately the following composition to be returned to the tank room: H_2SO_4, 1100; As, 1; Sb, 0.2; Fe, 1; Ni, 5.3, and Zn, 1.5 g. per liter.

The Cu, As and Sb deposited in the insoluble-anode tanks are in the form of a black slime, some of the slime adhering to the cathode but the greater portion settling to the bottom of the tanks. An average assay of this slime is given in Table X.

TABLE X.—SLIME FROM INSOLUBLE-ANODE TANKS

(Treating electrolyte direct from tank room)

Moisture, per cent	10.0
Cu, per cent	55.1
SiO_2, per cent	1.1
FeO, per cent	0.4
Al_2O_3, per cent	0.4
CaO, per cent	0.3
S, per cent	4.1
As, per cent	10.3
Sb, per cent	2.5
Ni, per cent	0.35
Zn, per cent	0.32
Ag, oz. per ton	3.4
Au, oz. per ton	0.02

This material was sent to the blast-furnace plant for treatment.

The method of purifying the electrolyte now in use is a modification of the one just described and consists in running off daily from the tank house about 25,000 liters of solution, concentrating this, by boiling, to 48° Bé. From the boiling tank the concentrated solution is run to crystallizing tanks and allowed to stand for four days. At the end of this period 82 per cent of the copper will have crystallized out. The mother liquor has the following analysis: Acid, 475; Cu, 17.4; As, 20.2; Sb, 1.1, and Fe, 15.2 g. per liter. This is run to the insoluble-anode tanks where it is treated for the removal of Cu, As and Sb as described above.

TABLE XI.—ANALYSIS OF INSOLUBLE-ANODE TANK SLIME

(Treating mother liquor from crystallizing tanks)

Moisture, per cent	9.66
Cu, per cent	46.30
SiO_2, per cent	0.38
FeO, per cent	1.66
Al_2O_3, per cent	0.4
CaO, per cent	1.08
S, per cent	5.02
As, per cent	21.48
Sb, per cent	2.28
Ni, per cent	0.35
Zn, per cent	0.32
Ag, oz. per ton	3.61
Au, oz. per ton	0.03

Table XI gives an analysis of the slime obtained when using this method.

After treatment in the insoluble-anode tanks the solution is either returned to the tank room or treated, by further concentration and crystallization as described above, for the removal of the Fe, Ni and Zn.

The analysis of slime from the second concentration and precipitation for the removal of Fe, Ni and Zn is as follows: Moisture, 22.1; S, 16.6; Ni, 5.6; Cu, 1.3; Fe, 10.3; Zn, 2.7 per cent.

The advantage of crystallizing the copper from the solution before treatment in the insoluble-anode tanks is that the ampere efficiency of these tanks is greatly increased because of the concentrated solution treated and that the amount of slime produced is reduced by nearly one-half, as shown in Table XII.

TABLE XII.—PRODUCTION OF INSOLUBLE-ANODE TANK SLIME

	Wet Slime per Month Lb.	As Per Cent.
Treating electrolyte direct from tank room	38,333	10.30
Removing 82 per cent. of copper before treating	21,600	21.48

The slime from the insoluble-anode tanks is sent to the blast furnaces for treatment, the iron slime is washed for the removal of free acid and sent to the blast furnace or wasted if free of copper.

The arsenic liberated from the anodes in the refining tanks finds its way into the following products in the proportions shown: In cathodes, 1.22; in silver slimes, 17.19, and in purification slimes and cathodes, 81.59 per cent. It is therefore necessary to remove from the electrolyte 81.59 per cent of the arsenic liberated at the anode.

Electrolytic Slime

From a converter anode carrying 40 oz. silver and 0.24 oz. gold per ton, a slime carrying 43.3 per cent Cu, 5000 oz. silver and 34 oz. gold per ton is obtained. See Table I for complete analysis.

This slime is removed from the refining tanks once in 60 days in the following manner: A copper jumper, held by the iron studs provided, connects the two ends of tank bars on adjacent tanks and short circuits four tanks, as shown in Fig. 2. The anodes and cathodes are removed and the slime, 5.5 in. having accumulated, is diluted with water and pumped by means of a bronze steam injector to a round lead-lined tank, 12 ft. in diameter, 4 ft. deep, situated 200 ft. distant in the slime-sampling room. Before entering this tank the slime passes through a lead screen with ⅛-in. holes and ½-in. centers. In this tank the slime is washed with hot water to remove the free acid and soluble copper.

From this tank the slime is dropped into a cast-copper, lead-lined egg from which it is forced by compressed air at 100 lb. pressure into a modified Bushnell filter press containing hard-copper plates and rings, 8-oz. duck being used for filter cloth. Thirteen cakes 26 in. in diameter and 1.25 in. thick are made at each charge; the wet weight is 865 lb. per charge at 21 per cent moisture. From the filter press the slime is placed on steam-jacketed copper-drying tables where the moisture is reduced to 10 per cent. The slime is then crushed with a roller on a cement floor and sampled in 22,000-lb. lots for shipment.

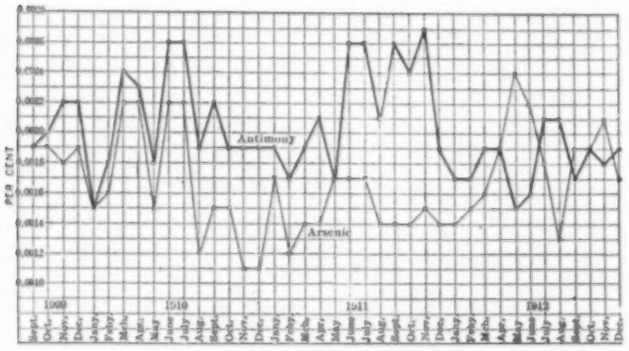


FIG. 8—RELATION OF ARSENIC TO ANTIMONY IN WIRE BAR

The slime is sampled in the following manner. The 22,000-lb. lot is shoveled into a conical pile, each shovelful being delivered upon the apex of the cone. To insure thorough mixing of the material this operation is repeated twice. The material is then split-shoveled four times; these operations reduce the sample to about 300 lb. From this point the material is quartered down with repeated crushing to about 130 oz., the laboratory sample.

The slime is then packed in 10-oz. canvas sacks holding 140 lb. each. Four of these sacks are packed in paper-lined wooden boxes in which shape the slime is shipped to an Eastern refinery for treatment. About 560,000 lb. of slime are produced and shipped per year.

Operating Details

The main tank room is divided into 10 divisions of 32 tanks each. Each division is taken care of by two tank men who are held responsible for these tanks. One inspector, locating and removing short circuits between anodes and cathodes, is employed for each two divisions. The inspector's work is done on night shift.

Sixteen tanks of two-day cathodes, per division, or one-half of the tanks, are drawn daily, each tank containing 22 anodes and 22 cathodes. The day's starting sheets having been delivered to the different divisions by the night shift, the tank men begin drawing cathodes in the morning. Four cathodes are drawn at a time with a 1-ton Yale & Towne triplex chain block and multiple hook. The cathodes after being replaced with starting sheets are deposited in lead-lined wheel carriages, each carriage holding 66 sheets. The solution dripping from the cathodes is held in these carriages to be later drawn off through an outlet in the bottom. When the carriage is loaded it is moved by hand to the scales, situated as shown in Fig. 3. After weighing the cathodes are lifted from the carriage by the traveling crane. The ends of the angle irons on which the cathode rods rest in the carriage are punched to receive the crane chain hooks by means of which the crane raises the load. After the cathodes are removed the carriage is weighed to obtain the tare. The crane then raises the cathodes and loads them on to a 4-wheel truck for delivery to the furnaces.

After the cathodes are drawn the tank men inspect the anodes, marking such as are ready to be removed as scrap. The same carriages as were used for the cathodes are now spotted under the trolley track at the end of the tanks and the anode scrap is deposited in them. When all of the scrap has been removed in this manner the carriages are moved to the scales and weighed. The crane then lifts the scrap from the carriages, by means of the angle irons, and conveys it to a tank where the slime is washed from the surfaces with a stream of water. The scrap is then placed on trucks, by the crane, from which it is loaded into box cars for shipment to the converters.

After the scrap has been all drawn from the tanks and the carriages removed the tank men proceed to fill the vacancies, thus made in the tanks, with new anodes, one anode at a time being handled.

As but four cathodes are removed from one tank at a time it is not necessary to cut the tanks out of circuit while drawing copper.

As compared with the traveling-crane system of handling the anodes and cathodes in tank units, the hand chain-block system of handling, as employed here, admittedly required more labor but is not without its advantages, which are as follows:

First, the chain-block system permits the use of converter anodes.

Second, the chain-block system, with aisles between the refining tanks, permits of starting sheets being straightened and placed in the tanks in such a manner that subsequent removal and straightening is not necessary.

Third, respacing of anodes and cathodes, from time to time, as the anodes corrode, to reduce the resistance of the tank, is unnecessary as the tank men properly space each separate anode and starting sheet as placed in the tank.

Fourth, there is a lower percentage of scrap and no scrap to be returned to the tanks.

Cash prizes are awarded to the tank men on the divisions making the best and second best records for the month. The division producing the highest tonnage of cathodes, after deducting the weight of the anode scrap, is considered as having the best record. This plan has been found very effective in stimulating rivalry between the different divisions and thereby improving the efficiency of operations.

Four insoluble-anode tanks to take care of the increase of copper in solution are provided. These tanks, as shown in Fig. 3, are situated in a separate room south of the pump house and, by means of a system of circulating pipes, can be run on any of the three electrolytes. It is not usually found necessary to operate more than two of these tanks at a time.

Four-day cathodes at 34 amp per square foot are produced in these tanks. The product of these insoluble-anode tanks is of the same purity as the product of the soluble-anode tanks provided the rate of flow of the electrolyte is sufficiently high to insure ample copper in solution in the lower tanks. When the rate of flow is greatly reduced or altogether stopped the electrolyte soon becomes impoverished in copper and arsenic is deposited out of the solution with the copper. The circulation is maintained in these tanks at the rate of 10 gal. per minute.

Table XIII is a summary of the average amount of copper, silver and gold locked up in the process while drawing two-day

cathodes and refining at the rate of 174,000 lb. copper per day.

To arrive at the weight of copper in the refining tanks, for inventory purposes on the first of the year, a spring balance is attached to the chain-block hook and the anodes and cathodes are raised clear of their supports and weighed in solution. A factor, previously determined, representing the difference in weight of the average electrode, weighed in solution in this manner, and its actual weight, is applied to the weights thus obtained. Weights obtained in this manner are found to check actual weights very closely.

TABLE XIII.—METALS LOCKED UP IN PROCESS

	Copper, Lb.	Silver, Oz.	Gold, Oz.
In anodes	2,300,000	44,000	316
In slime	22,300	140,000	850
In cathodes	180,000	.....	.....
In solutions	95,000	.....	.....
Totals	2,597,300	184,000	1,166

The amount of slime in the bottom of the refining tanks at any time is calculated from the tank cleaning record, the number of days the slime has been accumulating and the amount of slime that is liberated at the anode per tank per day per ampere being known.

Current readings are taken and recorded hourly from a 10,000-amp Weston station ammeter the shunt of which is situated in the center of the tank house. At the power house on each generator-switchboard panel is a 5000-amp instrument of the same type. The averages of the hourly readings from these instruments are compared daily with the readings of the tank-house instruments to detect possible current leakage.

A system of pressure wires connects the refining tanks and the generators with a voltmeter switchboard situated in the office at the electrolytic plant. Hourly readings of the voltages at generators, at the tank house and at the different divisions and subdivisions of refining tanks are taken and recorded. In this manner abnormal conditions in any group of refining tanks or any change in resistance of the transmission line are promptly made known.

The daily report contains, on one sheet, the cathode production for the day and to date for the month, the number and weight of starting sheets used, the pounds of copper deposited per tank per day per ampere in the different sections, the percentage ampere efficiency of the total production, the amount and percentage of anode scrap for the day and to date for the month, the average amperes, volts and kilowatts for the day and for the age of the cathodes drawn, the average weight per cathode drawn, the pounds of anodes, cathodes and anode scrap on hand, the pay roll for the day and a digest of the electrolyte men's reports showing amount of solution treated in the purifying plant, character and speed of circulation of electrolyte and other minor data.

The present practice is to draw three-day cathodes on Mondays and Tuesdays, two-day cathodes on the four days following and none on Sundays, the generators running continuously during the seven days.

Under the head of 22 anodes and 22 cathodes per tank in Table V, the results obtained are shown in detail.

The average daily production of refined copper is 174,000 lb.

Furnace Refining

The refining-furnace plant, to be operated in connection with the electrolytic refinery, began producing anodes from converter copper in October, 1892.

The original installation consisted of two coal-fired reverberatory furnaces. These furnaces were identical in design, having a hearth 15 ft. 6 in. x 10 ft. 6 in. with a firebox 4 ft. x 4 ft. Each furnace was provided with a separate brick stack 65 ft. high, 28 in. x 28 in. inside. The capacity of these furnaces was 30,000 lb. of copper each.

The furnace building was 80 ft. x 128 ft. steel frame covered with corrugated iron.

During 1893 and 1894 two additional furnaces of the same type and capacity were added. Two of the furnaces were employed in producing anodes from converter pig and anode scrap from the electrolytic plant and two were used to make wire bar, cake and ingot from the cathodes.

All of the furnaces were dipped by hand, 9-in. ladles being used. The copper was cast in iron molds, the electrolytic copper being dumped in boshes containing water. The anodes were removed from the molds, by means of hooks, and allowed to cool in the air.

Later these four furnaces were replaced by two larger furnaces of an estimated capacity of 50,000 lb. of copper each, subsequently increased to 100,000 lb. each. The hearths of these furnaces are 14 ft. x 24 ft. and the fire boxes are 7 ft. long by 8 ft. wide. One of these furnaces is connected to one of the original brick stacks, described above, and the other has a new brick stack 74 ft. high with a 34-in. x 34-in. flue.

These two furnaces with a combined capacity of 200,000 lb. per day now handle the cathode output of the electrolytic refinery, the anodes coming direct from the converters.

The product of the refining furnaces is dipped into copper molds, made at the furnaces. These molds are hung in trunnions over cast-iron boshes through which cold water is circulated. The mold is capsized, as soon as the metal is set, and the copper drops into the cooling water. Cooling in this manner prevents the formation of cupric oxide on the surface of the castings.

The copper is conveyed from the furnaces to the molds by means of large ladles suspended by chains from a trolley running on a ¾-in. x 4-in. track suspended over the boshes. Hand-forged Welsh ladles of the following sizes and capacities are used: 14-in., capacity, 200 lb. of copper; 16-in., 250 lb. of copper, and 19-in., 350 lb. of copper. The size of the ladle used is governed by the weight of the casting to be made.

The principal forms into which the copper is cast are as follows: Ingot, ingot bar, cake, wire bar and a special form of round billet used in the manufacture of seamless copper tubing. The copper is shipped direct to the consumer. A 100,000-lb. charge is taken out of each furnace once in 24 hours.

The usual practice of rabbling and poling the copper, for the oxidation of impurities and bringing of the metal to the proper pitch for casting, is carried on. The rabbling is effected by introducing compressed air at a pressure of 16 lb. per square inch into the molten bath by means of two ¾-in. iron pipes, the end of the pipes being kept below the surface of the metal. About two hours is required for this operation. The reducing action is obtained by forcing green pine poles into the bath and covering the surface of the metal with charcoal. About 28 poles, 8 in. at the butt and 4 in. at the top, and 35 bushels of charcoal are required per charge of 50 tons of copper.

The condition of the bath, as regards oxygen contents, is noted from time to time by the refiner as he examines the successive button samples taken from the bath in a small ladle. When the physical appearance of the button indicates that the "tough-pitch" stage has been reached the poles are removed and the dipping operation is begun.

As the dipping proceeds the refiner observes the set of the castings made and adds charcoal or logs of wood to maintain the oxygen at the proper point.

The skimming of the furnace takes place just before the rabbling or oxidizing is started. The slag obtained is equivalent in weight to about 3.5 per cent of the upper charged, varying with the amount of lime used. The following is the average analysis of slag produced at furnace No. 2 producing wire bar: Cu, 62.89; SiO_2, 19.1; FeO, 2.0; Al_2O_3, 1.8; CaO, 7.2; MgO, 1.4; S, 0.17; As, 0.02; Sb, 0.025 per cent; Ag, 0.350 oz. per ton.

The fuel used in these furnaces is known as Lochray lump coal, a bituminous coal mined at Tracy, Mont., 15 miles from these works. The analysis is as follows: Moisture, total, 7.01; moisture, combined, 1.7; volatile matter, 24.7; fixed carbon, 48.0; ash, 18.5; sulphur, 3.5 per cent; B.t.u. per pound of dry coal, 10.803.

It will be noted that this coal is high in sulphur. To protect the copper as it melts from the sulphur dioxide gases, resulting from the combustion of the fuel, about 30 per cent of the cathodes going to make up wire bar charges are dipped in milk of lime before charging. As the copper melts the lime forms a protective covering for the metal, hindering in a large measure the absorption of sulphur by the molten copper. This whitewashing process accounts for the lime and magnesia in the furnace slag.

The average analysis of wire bar produced at these furnaces is shown in Fig. 5, and the relation of arsenic to antimony in wire bar is shown in Fig. 8. The analysis of cake, ingot and other furnace products, in which high electrical conductivity is not essential, will contain 0.02 in 0.05 per cent less copper, due to the increase in oxygen contents.

Copper Deoxidation by Means of Magnesium [1]

By Dr. Friedrich Hüser

Copper has a great affinity to oxygen, and a still greater one to sulphur. The affinity increases with increase of temperature. Copper absorbs chiefly oxygen from the furnace gases, but also some sulphur dioxide[2] and hydrogen, while nitrogen, carbon dioxide and carbon monoxide[3] are not absorbed. The three first-named gases dissolve in liquid copper during the oxidizing smelting process.

When the liquid copper solidifies, part of the hydrogen forms a solid solution, while oxygen and sulphur separate as cuprous oxide and cuprous sulphide.[4]

Since at a high temperature the affinity between sulphur and copper is smaller than between sulphur and oxygen, there will be a reaction between cuprous oxide and cuprous sulphide, liberating sulphur dioxide:

$$2Cu_2O + Cu_2S = SO_2 + 6Cu.$$

Refined copper is supposed to contain only traces of sulphur. Nevertheless the evolution of sulphur dioxide may become troublesome when copper solidifies, since cuprous oxide and cuprous sulphide form eutectics with copper and finally solidify as such. The solidification of the pure copper begins from the wall of the mould so that the eutectics are forced to the center, forming there an enriched portion. A more vigorous reaction sets in, which is shown by the rising of the metal. The copper becomes porous. Naturally particles of these eutectics remain stuck between the growing copper crystals, as shown in Figs. 1 to 3 and 5.

When poling copper the sulphur dioxide dissolved in it is mechanically removed and the cuprous oxide is reduced, with the exception of a small amount.[5] Schnabel[6] says: "The porosity of copper may also be avoided in other ways than by means of poling, by reducing the cuprous oxide by means of substances which do not develop gases during reduction. Phosphorus, phosphor-copper and manganese-copper are such substances."

It may be said in general: If to melted copper additions are made of other more easily oxidizable elements like Sn, Zn, Ni, Fe, Mn, P, Si, Al, Mg, the whole of the cuprous oxide present is theoretically reduced with formation of the oxide of the other metal. However, on account of the great dilution in the solvent (copper in this case) an excess of the reducing agent is necessary. This excess must be the greater the smaller the tendency of the reducing agent to form oxides. If this excess remains in the copper it may prove deleterious.

In the series of metals mentioned above the affinity to oxygen increases from the left to the right.

Magnesium is the metal furthest to the right. It has the strongest affinity for oxygen. Small additions of it suffice to reduce any cuprous oxide remaining in molten copper. The addition of ⅛ per cent., which has been mentioned (Electrician, vol. 59, 1907, p. 463), is arbitrary; the amount of magnesium required depends, of course, in each individual case on the content of cuprous oxide which is to be reduced.

I have made a number of melts in a crucible, which showed that copper containing oxygen can always be refined by means of magnesium. The starting product used in these experiments was electrolytic copper of great purity. It was free from tin, iron, lead, arsenic, sulphur, and contained 0.01 per cent. zinc, a trace of nickel and 0.0237 per cent. of oxygen. Six to 10 kilograms of this copper were melted for each experiment in a graphite crucible, in a coal-fired furnace working with forced draught.

The required amount of magnesium varied with the speed of melting and also with the length of time for which the melted copper remained in the furnace after the melt was over.

If the melting took place very rapidly under a charcoal cover, a few hundredths of a per cent. of magnesium were sufficient. An alloy of magnesium and copper, containing 20 per cent. of magnesium, is preferable to more magnesium for these experiments. Its use prevents a too violent reaction and losses in manipulation. This was of some importance, on account of the small quantities that could be melted in the crucible. This alloy is brittle and may be weighed very conveniently. For refining on a large scale the introduction, without loss, of pure magnesium into the copper bath will prove less difficult.

The addition of an acid slag, finely powdered and easily fusible, after the deoxidation had taken place, proved very advantageous in these melts. This slag melted very rapidly and by vigorous stirring of the contents of the crucible it was possible to absorb all the magnesium oxide formed. In this way pure copper was always obtained.

In the beginning I used an iron calcium silicate slag (of the composition $FeO.CaO.2SiO_2$); in later experiments a cupola slag of similar composition served the same purpose. An addition of quartz in the shape of building sand made the slag more liquid, when it had taken up much oxide.

The illustrations show two copper samples. Figs. 1 to 3 show a copper that had been melted in a crucible and was cast into a cylindrical mould of 60 mm. diameter. The electrolytic copper mentioned previously was used as starting material. The melt was made without any addition for refining. As the copper had a chance to absorb some of the furnace gases, it was very turbulent during casting and rose in the mould. The bar shown in the illustrations was cut lengthways, polished and etched with cuprous ammonium chloride. It was 600 mm. in length. The illustrations represent the bottom end, the top end and a cut through the center.

The whole bar is porous. When crystallization has started it has been disturbed by the development of gas. A strong tendency toward dendritic formation (pine-tree crystals) is noticeable all over the surface. The oxygen content of the copper after melting was

$$0.298\%\ O_2 = 2.67\%\ Cu_2O.$$

The microphotograph of Fig. 7 (× 75 enlargement) corresponds to a highly poled copper. The darker portions represent cuprous oxide, which, mixed with copper, has the eutectic composition. The dendritic architecture of the copper crystals can also be seen.

The same quantity of electrolytic copper as used in the previous experiment was refined after melting by the addition of 0.1 per cent. magnesium. The bar cast from this melt underwent the same treatment as the one represented in Figs. 1 to 3. The results obtained with the magnesium treatment are shown in Figs. 4 to 6.

At the top (Fig. 4) a deep depression is shown, thus proving the absence of gas. The etching shows well-formed crystals. The varying luster of the crystals is due to their varying orientation toward the surface.

The corresponding Fig. 8 is a microphotograph (75 × en-

[1] Translated from *Metall und Erz.*, May 22, 1913.

[2] Schenck and Hempelmann, *Metall. und Erz.*, vol. 1, 1913, p. 283.

[3] Sieverts and Krumbhaar Zeit. f. Phys. Chemie, vol. 74, 1910, p. 3.

[4] Heyn and Bauer, Zeit. f. Anorgan. Chemie, vol. 39, 1904, p. 1, and Metallurgie, vol. 3, 1906, p. 73

[5] See the photograph in *Metallurgie*, vol. 19, 1909, p. 609.

[6] Schnabel, Handbuch d. Metallh., 1901, vol I, p. 270.

largement) and shows only very small amounts of inclusion in the copper.

The copper refined in this way and cast has a breaking strength of 18.5 to 21 kilograms per square millimeter, with an elongation of 30 to 37 per cent. It is very ductile and can be rolled into very thin sheets without tearing at the edges.

Table I shows the comparative results obtained when various substances were used as refining (reducing) agents in the melt.

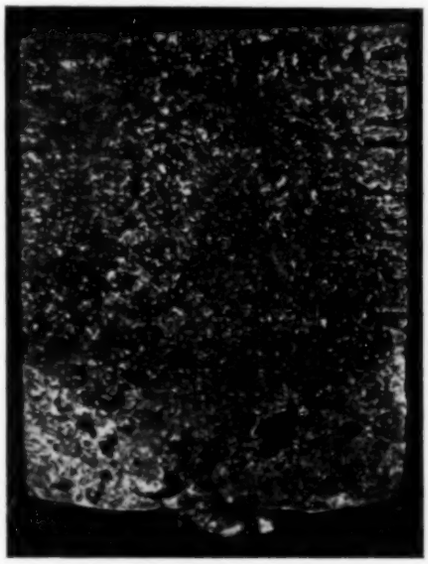


FIG. 1.—TOP SECTION (WITHOUT MAGNESIUM TREATMENT)

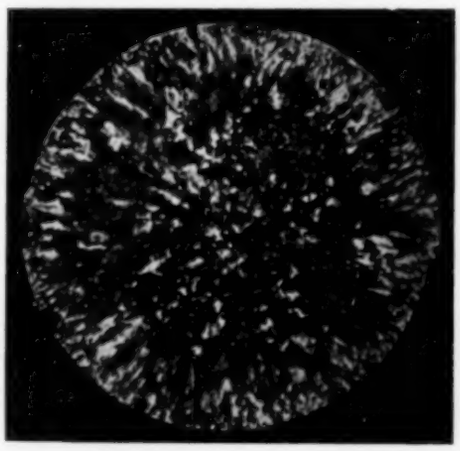


FIG. 2.—CENTRE SECTION (WITHOUT MAGNESIUM TREATMENT)

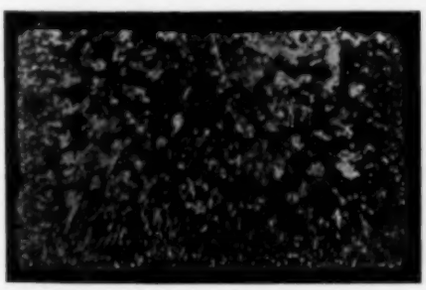


FIG. 3.—BOTTOM SECTION (WITHOUT MAGNESIUM TREATMENT)

Most of these belong to the series of elements given before.

For the melts I used those amounts of reducing materials, which, as determined by experiment, were required to produce a sinking of the copper when casting it. The residue remaining in the copper was determined later by analysis. This residue appears to be large in some cases in comparison with the amount added. This is due to the fact that a part of the oxide formed is retained by the copper and shows in analysis. The starting material was in all cases the above-mentioned electrolytic copper.

Each of the bars produced was rolled to a wire of 8 mm. diameter and then drawn to 5 mm. diameter. The wire was

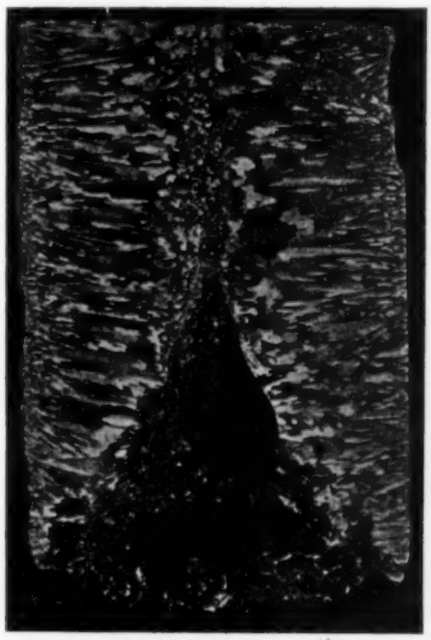


FIG. 4.—TOP SECTION (WITH MAGNESIUM TREATMENT)

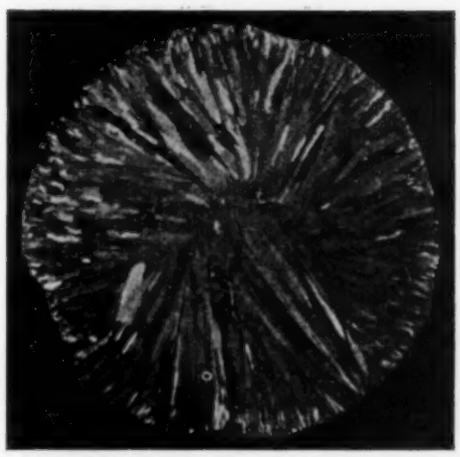


FIG. 5.—CENTRE SECTION (WITH MAGNESIUM TREATMENT)

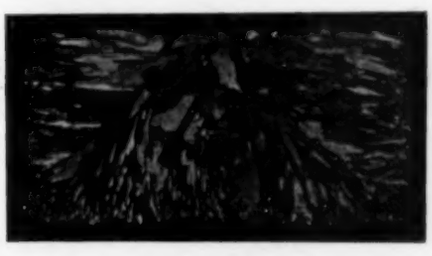


FIG. 6.—BOTTOM SECTION (WITH MAGNESIUM TREATMENT)

not annealed; it, therefore, shows high breaking strength and small elongation. Generally speaking, the breaking strength increases with an increase in the addition of the reducing agent, as was to be expected. If the wire is annealed, the breaking strength decreases by 15 kilograms and more, while the elongation increases correspondingly.

The most important figures in the table relate to electrical

TABLE I

Addition of refining material in per cent	0.4 Sn	0.4 Zn	0.2 Ni	0.2 Fe	0.2 Mn	0.2 Si	0.15 Cr	0.1 B	0.1 P	0.1 Al	0.1 Mg	0.1 Mg	0.1 Mg	Starting Material
Content of the material in copper in per cent	0.37 Sn	0.35 Zn	0.18 Ni	0.16 Fe	0.13 Mn	0.17 Si	0.04 Cr	.....	0.07 P	0.09 Al	0.013 Mg	0.024 Mg	0.039 Mg	
Electrical conductivity in per cent. of that of purest copper (100—60)	73.0	73.0	79.1	58.8	70.5	55.3	91.0	83.8	62.1	83.8	98.5	94.0	89.1	97.1
Breaking strength, kg. per sq. mm	50.1	44.5	40.8	45.5	43.3	42.6	39.3	42.6	44.4	41.7	42.9	41.4	41.9	42.8
Elongation in per cent	3.5	3.0	2.0	3.5	3.0	3.0	3.0	3.0	2.5	3.0	2.75	2.0	3.0	2.0
Number of twists, radius of 100 mm	7	8	8.5	6.5	7	7	7	8	8.5	7	8	7.5	6	5.5

conductivity. This is always diminished by foreign substances in the copper. Special attention is, therefore, necessary to see that nothing of these foreign substances remains in the copper after they have been used for deoxidation. Very small amounts depress the conductivity considerably, and their use

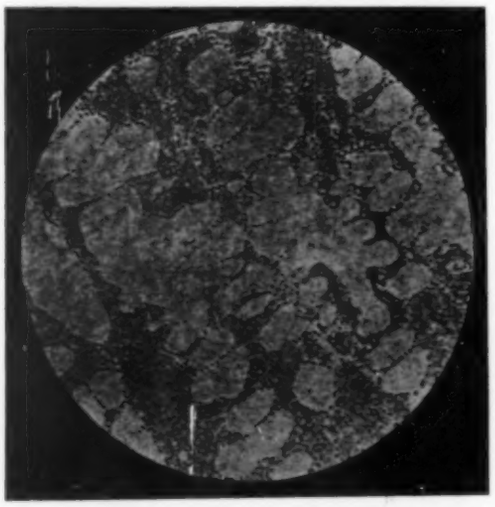


FIG. 7.—MICROPHOTOGRAPH (X 75 DIAM.) WITHOUT MAGNESIUM TREATMENT

for refining purposes is to be discouraged, if a high conductivity is required and a residue is unavoidable.

The figures in the table are self-explanatory. The three samples refined with magnesium show the highest electric conductivity. In the first of these three samples the conductivity

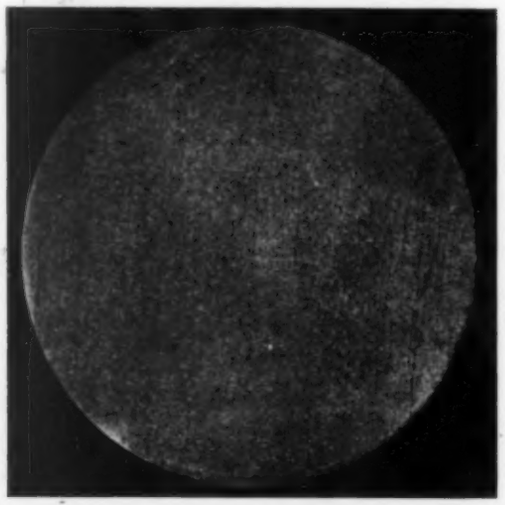


FIG. 8.—MICROPHOTOGRAPH (X 75 DIAM.) WITH MAGNESIUM TREATMENT

is higher than that of the starting material, although the latter was very pure and was considered to be of the highest conductivity.

It is to be noted that, though the amount of magnesium added was the same in all three cases, the degree of purity differs for the three melts. This must be due to varying amounts of Cu_2O at the time of refining. It is difficult to determine the instantaneous degree of purity in small melts of a few kilograms in a crucible. In the regular operation on a large commercial scale this is easily possible.

As a result of these experiments it may be said that magnesium is well suited for the refining of copper. If the object is to remove the last traces of oxygen in desulphurized poled copper or in melted electrolytic copper a few hundredths of one per cent. of magnesium are sufficient. It may be assumed that when the refining by means of magnesium is carried out on a large scale with the use of a slag and simultaneous poling, the magnesium oxide will rise very quickly to the surface of the bath.

To sum up, the comparative experiments on the deoxidation of copper with different reducing agents show that the best quality of copper, especially the highest electrical conductivity, was obtained when using magnesium. The use of a bi-silicate slag for absorbing the resulting oxide proved very advantageous.

The Canadian bounty on lead ores has been extended until 1918, or until the remaining available sum of $600,000 is expended. The bounty was originally established in 1903, at which time an appropriation of $2,500,000 was made to stimulate lead production in the Dominion. The rate of the bounty is based on the price of lead, decreasing as the market price rises.

The Buckhorn cyanide mill in Eureka county, Nevada, is nearing completion as rapidly as possible. All the concrete foundations for machinery are in, and the crushing plant is practically finished. This property belongs to George Wingfield, and will be a thoroughly modern plant.

Registration and protection of patents and trade-marks in the Philippines has been provided for by an act of the Philippine legislature. Patents registered in the United States Patent Office, and then filed in the Executive Bureau of the Philippines, will receive the same protection as is accorded in the United States.

Cyanide will be placed on the free list if the Senate committee has its way in the revision of the tariff. The House proposed a tariff of 1½ cents per pound, which is 2½ cents lower than at present. It is uncertain what the outcome will be when the matter is considered in conference.

The University of Utah School of Mines is a beneficiary of the late James McGregor to the extent of $50,000. Mr. McGregor was a mine owner in Utah, and interested in mining education. The fund is to be known as the James McGregor University of Utah Endowment Fund, and the income is to be used as the interests of the school may require. The Rose Polytechnic Institute, Terre Haute, Ind., was residuary legatee, and will receive about $200,000.

Ferronickel and ferrochrome are to be produced electrically by a French company which is constructing hydroelectric power plants and electric smelting works in New Caledonia. The project is to cost over $1,000,000, and will not be completed for two years.

The value of Montana metal production in 1912, according to the United States Geological Survey, was $64,754,613, against $46,955,287 in 1911. The increase was due largely to the greater production and higher price of copper.

Montana Meeting of the American Inst. of Mining Engineers

An Account of the Proceedings of the Eminently Successful Meeting Held at Great Falls, Anaconda and Butte, with Extracts of Papers

The 105th meeting of the American Institute of Mining Engineers was opened on Saturday, August 16, at Great Falls, Mont.

Saturday—Great Falls

One hundred members and guests registered on Saturday morning in Great Falls. The falls and hydroelectric power plants were visited in the morning, the Boston and Montana smelter in the afternoon, while in the evening a professional session for the presentation of papers was held in the Court House.

C. W. Goodale, for the local committees, made an address of welcome to the visitors, whereupon papers were presented by M. Hebgen, W. T. Burns, C. W. Goodale and J. H. Klepinger, E. E. Brownson, A. E. Wheeler and M. W. Krejci, D. A. Lyon and R. M. Keeney. Very great interest was manifest in the big hydroelectric developments and the possible uses of electric power for metallurgical purposes. Abstracts of the papers presented are herewith given.

Hydroelectric Development in Montana

The paper by **Max Hebgen,** Butte, Montana, on this subject first outlined briefly the natural features of the state of Montana affecting power development and then reviewed early water-power developments at the Big Hole plant, Canyon Ferry plant, Madison plant No. 1, and Black Eagle Plant, and the later developments of the Madison river system, the Missouri system, the Great Falls system, the Missoula river development, and the Big Fork development.

The total present generating capacity of the fourteen plants now operating on the combined systems of the Montana Power Company and the Great Falls Power Company is 72,100 kw (of which the capacity of the Rainbow plant is 25,000 kw and that of the Hauser Lake plant 14,000 kw). The principal transmission lines of the combined system operate at 46,200, 66,000 and 100,000 volts.

The paper concluded with notes on the uses of this power for lighting, traction, mining, and irrigation, and data on new power developments and undeveloped powers. The larger possibilities for developing new powers exist in the northwestern part of the state of Montana where developments like the Flathead lake, Kootenai falls and other large streams present possibilities for the development of at least 250,000 hp.

Great Falls Electrolytic Plant

The paper by **Willis T. Burns,** Great Falls, Mont., on this subject, is printed elsewhere in this issue, practically in full.

Great Falls Flue System and Chimney

A long paper by **C. W. Goodale,** Butte, Mont., and **J. H. Klepinger,** Great Falls, Mont., describes in detail the new flue system, completed in 1909 at the Boston and Montana Smelter at a cost of $1,100,000. It includes a chimney 506 ft. high and 50 ft. inside diameter.

A comparison of the losses of copper, silver and gold, from the old flue system and from the new flue system shows that the new flue system has effected a saving of $372.18 per day, or $130,263 per year.

In addition to this saving it is expected that a substantial revenue will be derived from the treatment of the more arsenical dust, in the recently completed arsenic plant.

In considering the large expenditure for the installation of this elaborate flue system, it should also be borne in mind that a reduction in losses of flue dust was not the only object in view. The low elevation at which the gases were discharged from the old chimney was a source of annoyance from smoke complaints. The overtaxed condition of the old flues and chimney prevented the operation of certain furnaces at full capacity, and for the same reason the smoke and fume around the furnaces materially interfered with the efficiency of labor. Furthermore, dust chambers were inadequate and their poor arrangement made the handling of flue dust expensive and troublesome.

All these defects have been eliminated, and the anticipated efficiency of the new flue system has been fully realized.

Determination of Arsenic and Antimony in Copper

A paper by **E. E. Brownson,** Butte, Mont., dealt with the determination of arsenic and antimony in converter and electrolytic copper. It described the methods used in the laboratory of the Boston and Montana Smelter. The determination is finished in 24 hours.

Great Falls Converter Practice

A long paper by **Archer E. Wheeler** and **Milo W. Krejci,** Great Falls, Mont., gave many interesting details of Great Falls converter practice.

After a general chronological review of the principal events and changes of practice at the Boston and Montana Smelter, the authors gave more detailed accounts of mechanical manipulation, casting of finished copper, tamping of linings, early acid linings, acid linings with the use of ore, introduction of flux through the tuyeres, bottom blast, disposal of converter slag, development of different classes of converters (five classes being distinguished, successively increasing in size, the largest class V being now considered the standard for the plant), basic lining, number and size of tuyeres, cast copper tuyeres, efficiency of air, size of converter mouth, height of converter.

The chief conclusions of the authors are as follows:

The basic lining has shown its superiority over the acid lining and therefore the acid lining can be dismissed as not being the best practice.

A large converter is better than a small one. There is no practical limit to the size of a converter which will work metallurgically. The limits are mechanical, and depend on the size of unit a plant will stand from the point of view of quantity of production, but even in this latter instance there can be a rather wide range of choice. The authors have found that the Class V (latest and largest size) or 20-ft. converter can be used as a storage for matte or for finished copper for long periods if the mouth is properly covered. Therefore in a plant of comparatively small production, a combination of ample matte storage capacity in blast-furnace settlers and reverberatory furnaces, together with a large converter, can be made so as to allow all converting to be done on one shift, and the casting of the copper perhaps to be done on the next shift.

With the success of the large units even the largest plants should have only a few large units and not a large number of small units.

From the experiments and observations which have been made, the authors feel that the tuyeres for any converter from 12 to 20 ft. diameter should be about 2.25 in. internal diameter. Tuyeres 2.75 in. internal diameter have not proved successful.

Tuyeres smaller than 2.25 in. diameter introduce undesirable construction features, particularly weakening the lining at the tuyeres, introduce unnecessary resistance to the entrance of the air, and add to the work of punching.

The mouth of the converter should be of ample size. Openings of 6 ft. 6 in. diameter for a 12-ft. converter and 8-ft. for a 20-ft. converter have not been found too large.

The bottom lining must not be too close to the tuyeres on account of a tendency of the bottom to build up and interfere with the punching of the tuyeres and also often causing a concentration of air in a few places with consequent throwing out of the charge. At least 5 in. should be allowed from the lowest point of the tuyeres to the bottom lining. There is no reason why this dimension cannot be increased, the determining factor being the size of charge it is desired to finish. There must always be copper enough at the finish to cover the tuyeres and a very deep bottom would require a large charge.

Air can be used efficiently in large volumes, for instance in the 20-ft. converter up to 22,000, or even more, cubic feet per minute. However, the best results seem to be for this converter about 18,000 cu. ft. while slagging.

As to the height of the converter, the Class IV, 13 ft. 8.5 in. high, and the Class V, 17 ft. 7.75 in. high, have proved very satisfactory with proper attention to the details of operation, and trials of a taller converter were not satisfactory. Probably the converter should not be made appreciably lower. The proper dimension to quote in this particular is from the center of the tuyeres to the lowest point of the mouth. These dimensions for the two classes are: Class IV, large mouth, 7 ft. 11 in.; Class V, 8 ft. 9.25 in. Class V is now considered the standard for the plant.

These the authors believe to be a few of the basic principles which have been demonstrated in practice.

Smelting of Copper Ores in the Electric Furnace

A paper by **Dorsey A. Lyon** and **Robert M. Keeney**, of the U. S. Bureau of Mines, Pittsburgh, Pa., discussed the smelting of copper ores in the electric furnace.

The authors first emphasize that the electric furnace was not developed as a competitor of the combustion furnace, but: (1) For the purpose of doing high-temperature work which it is not possible to do in the combustion furnace; and (2) for the treatment of ores from deposits which are located in regions where fuel is scarce and costly, and where hydroelectric power is comparatively cheap, as is the case in Chili, in Canada, in certain parts of this country and Mexico.

"This paper is not presented with the idea of trying to prove that the electric furnace should replace the reverberatory or the blast furnace as used at present in the smelting of copper ores, but that it may be substituted for them in those localities which are not favorable to them, and where either of the two following conditions may exist:

(a) "A district remote from railway transportation facilities, and where there is a deposit, or deposits, containing sufficient values to warrant their being worked, but which, due to the cost of getting coke in for operating a smelter and of getting ore out to a smelter, have to remain unworked. And where, on the other hand, there is plenty of water power which can be developed at reasonable cost for the production of electric power, by the use of which for smelting the ores in an electric furnace, with subsequent Bessemerization if necessary, a concentrated product may be obtained, which can easily stand the necessary transportation charges.

(b) "Or, comparatively speaking, the transportation facilities may be all right, but the cost of fuel be too great to permit of the ordinary methods of smelting, whereas, on the other hand, hydroelectric power can be developed in the district at a low cost, and the ores could be smelted by the electric furnace at a cost that would make their treatment possible."

The authors then take up the principles of the chemistry of copper smelting and concisely deal with the treatment of native copper ores, oxide and carbonate ores, and sulphide ores.

As to *reverberatory smelting*, the reverberatory furnace may be said to be used for the smelting of ores and concentrates which cannot be successfully treated in a blast furnace. If, as Peters states, the proper function of the reverberatory furnace, in smelting copper ores, is to generate heat as rapidly as possible, there is no apparent reason why copper ores cannot be smelted equally as well in an electric furnace, and perhaps even more advantageously, for the thermal efficiency of the average reverberatory furnace is only from 5 to 8 per cent., although about 20 per cent is now obtained in the large Anaconda reverberatories, whereas, that of an electric furnace which would be used for smelting copper ores suitable to refractory smelting, would probably have an efficiency of as much as 70 per cent. Chemically the only ill effects that might possibly result from the use of such a furnace would be the effect of the carbon of the electrodes upon the iron oxides which might be contained in the charge and which would result in:

1. The loss of electrodes.
2. The production of metallic iron, which would contaminate the metallic copper and increase the cost of refining; or, if a matte was the final product, might necessitate re-concentration.
3. The loss of electric energy, due to the additional heat consumed by the reduction of iron.

Since the reverberatory furnace is essentially a means of melting, whether used for sulphide, oxidized or native ores, the results obtained in the electric smelting of native copper ores will serve for the sulphide and oxide ores also, for comparison with the reverberatory in order to clear up the three points referred to above. The slags would be somewhat similar in each case. The atmosphere of the electric smelting furnace charged with sulphide ore would not be as neutral as in the furnace charged with native copper ore, because of the tendency of the elemental sulphur and any sulphur dioxide formed to dilute the air so as to make the furnace practically reducing as far as the consumption of electrode by oxygen from the air was concerned. If the electric furnace was not kept closed, considerable sulphur dioxide might be formed, which would possibly be reduced in part by the carbon electrode. Even with such reduction of sulphur dioxide, the electrode consumption would probably be less when working on sulphide ores than when working on native copper ores. This is shown by Bureau of Mines experiments, when, with a charge of native copper concentrate, the maximum electrode consumption in fourteen experiments was 48.0 lb. of graphitized carbon per ton of charge, the minimum, 22.1 lb. and the average 34.5 lb. In the same furnace, with charges of sulphide, oxide and siliceous ores mixed, a matte being the product, the maximum electrode consumption for 20 runs was 60.0 lb., the minimum, 10.8 lb., and the average, 23.3 lb.

SMELTING OF FINE MICHIGAN NATIVE-COPPER CONCENTRATES IN THE ELECTRIC FURNACE

For the purpose of investigating the feasibility of smelting fine Michigan native-copper concentrates in the electric furnace, instead of the reverberatory furnace, experiments were conducted by the authors. The objects of these experiments were:

(1) To note the effect of variation of the three main slag constituents, silica, ferrous oxide and lime, upon the power consumption, the operation of the furnace and the purity of the copper produced.

(2) To ascertain the extent of iron oxide reduction caused by the graphite electrode.

(3) To compare intermittent smelting with continuous smelting.

Two grades of concentrates were smelted: No. 3 and No. 4, of which No. 3 was much the coarser; of the former, 69.73 per cent passed through a 60-mesh sieve, as compared with 88.12 per cent of No. 4. The chemical analyses of the ore were as follows:

No. 3. 37.35 per cent Cu, 33.33 SiO_2, 15.40 FeO, 8.78 Al_2O_3, 5.18 CaO, 1.09 MgO, 0.056 S and 0.30 Fe (metallic)

No. 4. 25.35 per cent Cu, 25.92 SiO_2, 21.60 FeO, 12.40 Al_2O_3, 5.98 CaO, 1.09 MgO, 0.096 S and 0.50 Fe (metallic).

Fifteen experiments were performed in a Siemens type of electric furnace, having a capacity of 50 kw. The furnace was operated as a resistance furnace. The charge and products of a typical experiment were the following:

Charge		Black Copper		Slag	
	Lb.		Per Cent.		Per Cent.
Concentrate No. 3	22.0	Cu	98.59 (by difference)	Cu	0.15
Hematite	1.1	Fe	1.10	SiO_2	44.88
		S	0.29	CaO	6.92
		As	0.02	MgO	2.32
				FaO	29.80
				Al_2O_3	15.34

The author's conclusions from these experiments are as follows:

(1) Fine Michigan native-copper concentrates can be smelted in the electric furnace with the production of a good grade of black copper and without excessive losses of copper in the slag or by other means.

(2) The percentage of copper in the slag need not exceed 0.25 per cent., with a slag of proper composition. The loss in the slag should not exceed 0.50 per cent of total copper charged. Other losses should not exceed 1.0 per cent; giving a total loss of 1.5 per cent of all the copper charged.

(3) The loss of copper by volatilization is high, if the slag is much more acid than a mono-silicate, and if it contains much alumina. The lowest losses were obtained by addition of limestone to form a slag containing 35 per cent SiO_2, 22 per cent CaO and MgO, and 25 per cent FeO.

(4) The difference between limestone and hematite as a flux is not marked, one being about as efficient for this purpose as the other, if the slag has not too high a specific gravity. No more iron was found in the black copper product when hematite was used than when limestone was used.

(5) With a furnace operating at a low temperature on a mono-silicate slag, the reduction of iron by the carbon electrode should not be excessive or sufficient to result in a product containing less than 95 per cent. metallic copper. With care a product containing 98 per cent. copper could probably be produced regularly.

(6) The total electrode consumption greatly exceeds the amount of carbon necessary for reduction of the iron found in the metal. An average of 9.35 per cent. of the total was used in iron reduction. The electrode consumption should not exceed 10 lb. per ton of charge.

(7) The electric furnace operates satisfactorily with the high siliceous alumina slags produced by the natural gangue without flux, but the losses by volatilization are high because of the high melting point of the slag. There is no copper oxide in the product, as was shown by microscopic examination.

(8) The power consumption for smelting fine Michigan native copper concentrates of between 25 and 40 per cent. copper in an electric furnace of 750 kw. or greater capacity should not exceed 640 kw. hr. per ton of ore smelted.

PROPOSED PROCESS FOR ELECTRIC SMELTING OF FINE NATIVE COPPER CONCENTRATES.

The commercial smelting of fine native copper concentrates in the electric furnace to attain the greatest economic success would be carried out in a continuous operation, consisting of charging the mixture of ore and flux at intervals into the top of an electric furnace having a short stack above the smelting crucible for preheating the charge before it reached the smelting zone, and tapping off the metal at intervals, the slag being permitted to run continuously from the furnace through a slag notch above the metal hole. The possibility of operation in this manner was shown in an experiment where there was an average of 1.10 per cent. iron in the product and 1.37 per cent. copper in the slag. The total loss of copper was as low as in the intermittent runs, and was largely due to the small scale operation. The slag loss was high because of imperfect settling, and because of the tapping of partially fluid material with the slag; a condition that would not occur in a large furnace operated properly. The power consumption was lower than in the other experiments.

The essential conditions for the successful performance of this process are a crucible, where the electrical energy is introduced and where the metal settles from the slag, and a short shaft above the crucible for preheating the charge before it reaches the smelting zone. As there is no reduction to be performed, this shaft need not be higher than to give plenty of head room between the top of the crucible and the charging floor for the moving of the electrode.

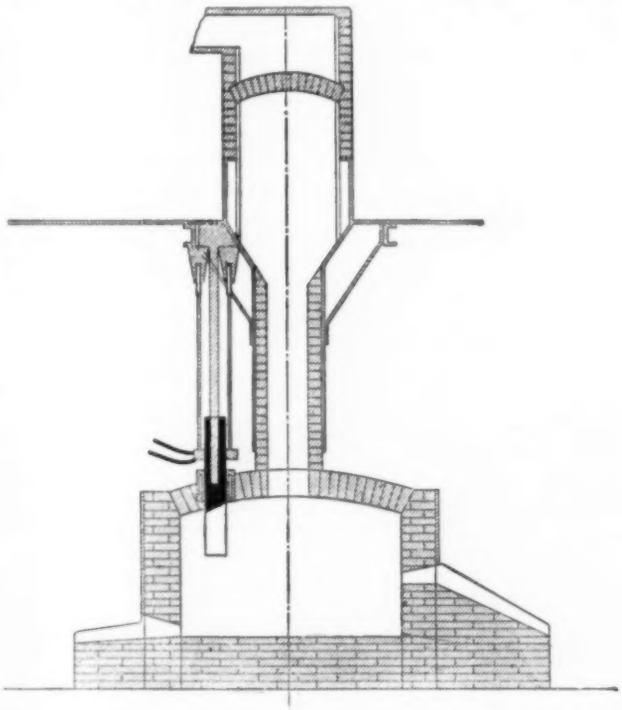


FIG. 1.—ELEVATION OF ELECTRIC FURNACE FOR SMELTING NATIVE COPPER CONCENTRATE

A furnace of the type shown in Figs. 1 and 2 satisfies these conditions. It consists of a smelting chamber lined with fire brick or silica brick, having three carbon electrodes suspended vertically through the roof with a feeding shaft lined with fire brick over the crucible. A 750-kw. furnace would treat, per 24 hr., about 23 tons of native copper concentrate, containing 25 to 40 per cent. copper. This furnace would be operated with three-phase current at from 50 to 100 volts. The general internal dimensions of the crucible are 14 ft. 6 in. in dameter, by 9 ft. high, with a shaft 1 ft. 6 in. in diameter by 18 ft. 6 in. high. With a furnace of this type regulation of electrodes would not be necessary after regular operation was established, so that the electrical connection is made close to the roof of the furnace and regulation is by hand. There is less difficulty in handling the electrodes if they are suspended vertically; and, as the shaft is low, it is made vertical, thus eliminating the necessity of suspending them at about 55 degrees.

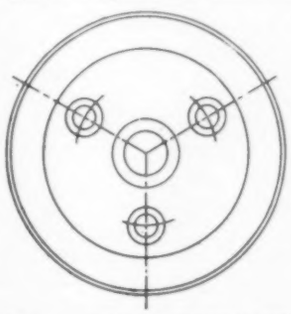


FIG. 2.—PLAN OF ELECTRIC FURNACE FOR SMELTING NATIVE COPPER CONCENTRATE

The slag from such a furnace should not contain over 0.25 per cent. copper, and could be discarded at once without further treatment. It could be granulated with water as discharged from the furnace. The metal would require further refining for elimination of iron and other impurities, which could be done by charging the hot metal directly into a re-

verberatory furnace, where it would be refined as previously described.

Briefly, the present process for treating these concentrates consists of the following steps: first melting and refining the coarse concentrate in either one or two reverberatory furnaces, from which a slag is produced containing from 10 to 30 per cent. copper; second, agglomerating the fine concentrate in a reverberatory furnace, or briquetting it with lime; third, smelting the slags of the reverberatory furnaces and the agglomerated or briquetted fine material, with coke in a blast furnace, the slags from which contain 0.6 to 0.8 per cent copper and the metal, 90 to 95 per cent copper; and fourth, retreatment of this metal in a reverberatory furnace.

The use of the electric furnace for treatment of these concentrates gives the following advantages: first, no agglomeration or briquetting would be necessary; second, no slags produced by this furnace would require retreatment; third, a large proportion of the higher-grade concentrates containing 75 per cent copper could be mixed with the fine concentrates and treated directly in the electric furnace, thus reducing the amount of slag necessary for re-smelting; and fourth, if cheap power was available the electric furnace could be used for reducing the copper from the reverberatory slags, and the resulting metal would probably be lower in iron and the slag lower in copper, because of the greater ease of regulating the reducing atmosphere, as only sufficient coke is present for the reduction of combined copper.

The limit to which the higher-grade concentrates could be mixed with the fine material would be determined by the electric conductivity of the mixture in the furnace. In the experiments performed with a charge containing 37 per cent metallic copper, no difficulty from short circuits or other causes was experienced. The point to which the amount of copper in the charge could be increased would have to be ascertained by experiment.

Technically the electric furnace does away with the necessity of briquetting or nodulizing, and, in general, facilitates handling the materials. The commercial factors influencing the cost would largely depend upon local conditions as to cost of power and fuel. Opposed to the cost of power would be the cost of fuel and briquetting or nodulizing, with possibly a higher labor cost for the present practice. If the cost of power was equal to these three factors, and no less, the electric furnace might still have the advantage of facility of operation. Also, in the case of a mine remote from smelting plants, it might be cheaper to concentrate the low-grade product of wet concentration by smelting in a small electric furnace, and thus reduce the cost of freight. Finally, to save losses of wet concentration, the ore might not be concentrated to so high a degree in this manner, but a low-grade concentrate made, which would be smelted directly in the electric furnace and the high-grade black copper shipped to the custom smelter.

We have obtained figures from a reliable source showing that the cost of treatment of these fine concentrates by agglomeration in a reverberatory furnace, refining of this copper produced, retreatment of slags in the blast furnace, and refining and casting the cupola copper into marketable shapes, is $8.64 per ton of 35 per cent native copper concentrate. This includes overhead charges, all repairs and maintenance of plant, but not amortization.

The treatment of the same material by the proposed electric furnace process to produce marketable shapes is estimated as follows:

Cost of Electric Power		Cost of Treatment Per Ton of Concentrate
Per Kilowatt Hour	Per Kilowatt Year	
0.5	$43.80	$7.18
0.625	54.80	8.08
0.75	65.80	8.87

A rate of one-half cent per kilowatt hour with a high load factor is possible in many districts with the use of steam turbine plant, if cheap hydroelectric power is not available. The load factor in this case would be above 90 per cent, and the power factor not below 0.85. In practice the cost of refining "cupola" block of the electric furnace would probably be lower than in the case of blast furnace cupola blocks, because of the higher percentage of copper in it. In the reverberatory process 81.3 per cent of the material charged must be retreated as slag in the blast furnace. In the electric furnace process, the only material to be retreated would be about 5 per cent as slag from the refining of the cupola blocks in the reverberatory. This slag could be retreated in the electric furnace. The great saving in retreatment charges accounts for the possibility of use of the electric furnace in this connection, even with expensive electric energy.

With respect to *ordinary copper-blast-furnace smelting* with carbonaceous fuel the author's first deal with the prerequisites of the process and the different losses and discuss whether any of the losses in a blast furnace could be avoided in an electric furnace. The dust losses, due to blast, would be less, although more or less fine particles would be carried away as dust by the gases escaping from the electric furnace. Volatilization losses would be less if the electric furnace would be operated more in the manner of a resistance furnace than as an arc furnace.

As to the losses in the slag due to particles of matte or metal, which have failed to separate properly from the slag, such losses would occur just the same if an electric furnace were substituted for the blast furnace, but there is no reason to believe that they would be any greater and on the other hand they should not be as large, for, due to the fact that it is possible to so control an electric furnace as to obtain the temperature desired, a sufficiently high temperature could be given to the matte and slag before it left the furnace to insure its being fluid enough to cause perfect separation in the settler, which would thus obviate this loss. As to the oxidation of valuable metals, this loss should be reduced to a minimum in a furnace operation without a blast, for the loss by oxidation is doubtless due to the action of the blast upon the products of the furnace as they pass the region of the tuyeres.

The authors summarize the experimental work which has been done before by others on the electric smelting of raw sulphide, oxidized and carbonate ores. This comprises Vattier's experiment (Report of Canadian Commission, 1904), Wolkoff's experiments (*Metallurgie*, vol. 7, p. 99) and Stephan's experiments (*Metall und Erz*, October, 1913; Metallurgical and Chemical Engineering, January, 1913, page 22).

Electric Smelting of Sulphide Ores

The authors then describe a series of experiments which they have carried out themselves at the Bureau of Mines Laboratory. The objects in view were: (1) to determine if there are any conditions arising which make electric smelting without air anything but a simple melting operation; (2) to note the percentage of concentration and the sulphur removal; (3) to study the possibility of condensation of the elemental sulphur as such; (4) to get general figures on power consumption with varying charges; (5) to determine gold, silver and copper losses, and (6), to study the use of a low-copper matte as a collecting agent for gold and silver.

1. *Ores Treated:* The ores used in the experiments were: A heavy sulphide low-grade copper ore; a gold; a silver, siliceous

	Pyrite, Per Cent.	Nodule, Per Cent.	Siliceous Ore, Per Cent.
SiO_2	2.50	4.00	74.53
Fe	44.07	65.60	10.71
S	48.20	....	8.36
Al_2O_3	0.24	1.76	0.40
CaO	0.08	0.45	....
MgO	0.73	0.46	0.26
P	0.02	....	....
As	0.20	....	....
Cu	1.30	0.07	....
Au	0.01 oz. per ton	0.03 oz. per ton	1.28 oz. per ton
Ag	0.14 oz. per ton	0.05 oz. per ton	3.20 oz. per ton

ore, and some roasted ore of the analyses shown below. The limestone contained 63.2 per cent CuO and 5.7 per cent MgO.

The ores were treated in the electric furnace described under the native copper experiments. The top was roofed and kept closed tightly to prevent escape of sulphur and admission of air. A condenser, consisting simply of three rectangular, horizontal chambers with several baffles in it, was attached to the furnace to condense the sulphur and catch any escaping dust.

Twenty experiments were made with these ores. A typical charge was as follows:

Charge		Matte		Slag	
	Pounds		Per Cent.		Per Cent.
Pyrite	8.8	Cu	1.22	Cu	0.05
Nodule	13.2	Fe	64.18	SO_3	35.25
Siliceous Ore	5.76	S	22.38	FeO	41.30
Limestone	2.73		Oz. per Ton	Al_2O_3	4.40
		Au	0.72	CaO	10.00
		Ag	0.96	MgO	1.64
				S	3.66
					Oz. per Ton
				Au	0.03
				Ag	0.32

2. Conclusions: (1) Smelting in an electric furnace treating sulphide ores consists simply of melting the charge, volatilizing about 60 per cent of the sulphur as elemental sulphur and separating the slag and matte.

(2) The ratio of concentration is simply that possible from a separation of matte and slag, together with a reaction of oxides and sulphates upon sulphides, and the volatilization of 10 per cent more than one atom of sulphur, but with no formation of iron oxide to enter the slag, because there is nothing present to oxidize the iron sulphide.

(3) Qualitatively, it is possible to condense some of the elemental sulphur driven off.

(4) The loss of copper, by volatilization and in the slag, is low.

(5) A matte containing about 1 per cent copper, from a charge containing 0.30 per cent copper, makes a good collecting agent for gold and silver, if a clean separation of slag and matte is obtained.

(6) There is very little loss by volatilization of gold in the electric furnace, but there is some loss of silver in this way. This would not probably occur in a larger furnace with closer temperature regulation.

(7) The electrode consumption in the smelting of a sulphide ore is low, and in practice need not exceed 5 lb. per ton of charge.

(8) In a large commercial furnace the power consumption for most ores would be about 480 kilowatt-hours per ton of ore, or 0.055 kilowatt-years.

ELECTRIC FURNACE VS. THE REVERBERATORY FURNACE AND BLAST FURNACE AS A MELTING AGENT

The authors believe to be justified in making the following statements as to the possibility of using the electric furnace for this purpose:

(1) The melting of native copper oxide or sulphide ore of copper can be done just as efficiently in the electric furnace, and perhaps more so, than in either the reverberatory furnace or the blast furnace.

(2) The reactions desired in reverberatory smelting, or ordinary blast furnace smelting, can be obtained in the electric furnace as well, and perhaps better, than in either of those furnaces.

(3) The loss of electrodes is small, varying from 5 lb. to 10 lb. per ton of ore melted, and the presence of the carbon electrode does not cause enough reduction of iron to be troublesome, or increase the consumption of electrical energy appreciably.

(4) The losses of copper, gold and silver by volatilization and in the slag would be no greater, as shown in all the experimental work cited, than they are in reverberatory smelting or ordinary blast furnace smelting.

(5) A matte containing as low as 1 per cent copper can be used as collecting agent for gold and silver in the electric furnace, as well as in combustion furnaces.

(6) The comparison of costs would depend entirely upon the nature of the ore treated, and the relative cost of coal or coke, and electrical energy. In general, from the work done in the experiments, we may say that from 500 to 700 kilowatt-hours per ton charged would be required for smelting copper ore, depending upon the nature of the ore.

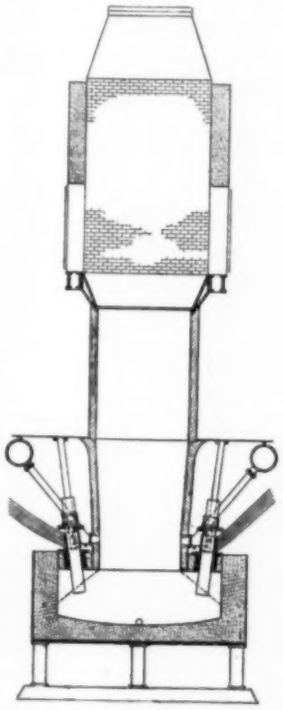


FIG. 3.—ELEVATION OF ELECTRIC BLAST FURNACE

SMELTING SULPHIDE ORES WITHOUT ROASTING, OR PARTIAL PYRITIC SMELTING

The authors finally take up semi-pyritic and pyritic smelting of sulphide ores, the object being to bring about the required degree of oxidation and the melting, at practically the same time, and to obtain either a whole or a part of the heat required for the process from the combustion of the iron and sulphur contained in the ore. As is well known, if practically all the heat necessary for the process is obtained in this manner, we have what is known as pyrite smelting. If only a part, semi-pyrite smelting.

The question, therefore, that presents itself in connection with the smelting of copper ores in the electric furnace is to determine whether electric heat may be used to replace the heat which is derived from the combustion of coke in pyrite and semi-pyrite processes. In other words, whether it would be possible, and if so, whether it would be commercially feasible, to carry out these processes in an electric furnace so constructed as to use a blast, thus obtaining as much heat as possible from the oxidation of the sulphur and iron, and supplying by electric energy whatever additional heat might be needed to carry out the process successfully.

The authors assume an electric furnace similar in construction to a modern copper blast furnace, and that part of the furnace, including the tuyeres, is practically identical with the same. Below the tuyeres, the furnace could be constructed as shown in Fig. 3. By referring to this it will be noted that there are electrodes extending down into the crucible, the arrangement of which, along the sides of the crucible, can be noted by referring to the plan of the furnace in Fig. 4.

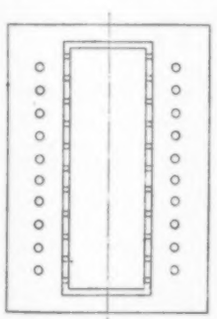


FIG. 4.—PLAN OF ELECTRIC BLAST FURNACE

The authors discuss at some length the possibilities of semi-pyritic smelting in an electric furnace and conclude that there does not seem to be any metallurgical reason why the chief objects of the pyrite smelter cannot be carried out in an electric copper blast furnace as well as in a coke copper blast furnace; namely, "to melt the great mass of SiO_2 and inert earths, to oxidize enough of the sulphides in the charge to insure a suitable matte—incidentally obtaining the heat evolved by the oxidation." On the other hand, it would seem that the difficulties ordinarily encountered in operating a pyrite furnace might be avoided when using an electric blast furnace, namely, the difficulties which arise when too much or too little coke is

added to the charge, and moreover that it not only permits "introducing heat into the furnace, without at the same time robbing the combustion zone of oxygen," of which Dr. Peters states, "nothing would be so welcome to the furnace man as to do this," but would also permit of the heat being entirely under control and easily regulated, and thus avoid freeze-ups with their consequent vexations and costly delays. By this the authors do not mean that any sort of a charge could be put through the furnace and not freeze it up, an idea which seems to be quite prevalent in regard to electric furnaces, but that, if the charge be properly calculated, a very much wider variation would be permissible in the composition of the slag from that calculated than would be the case in blast furnace smelting.

By referring to Figs. 3 and 4, it will be noted that the chief difference in the construction of the furnace would be in that part of it below the tuyeres—in other words, in the crucible. Because electric furnace construction has received the attention of some of the very best mechanical and electrical engineers, and because a crucible based upon the principle shown in the design in Fig. 3 is now extensively used in the iron electric reduction furnaces of Norway and Sweden, the authors believe that no difficulty would be experienced in this respect. Various methods and arrangements can be used for connecting up electrodes. If this be true, only the matter of costs has to be considered.

In figuring the cost, the authors assume that the furnace is to smelt 384 tons of charge per day that the charge contains 5.21 per cent Cu, 26 SiO_2, 19 FeO, 11 S, 4 Al_2O_3, 17 CuO. Such a charge has been smelted at the Washoe smelter with 8.2 per cent coke, or with 164 lb. of coke per ton of charge. If its calorific value was replaced by electric heat, the authors reason that taking into account the relative efficiencies of the furnaces, 400 kw. hours would be required to smelt one ton of charge. Since 384 tons are to be smelted per day or 16 tons per hour, 6400 kw. would be the constant load of the furnace.

"We may say that the costs of coke and electrical energy are about on a par when coke costs $7 a ton, and electrical energy 0.15 cents per kilowatt hour, or $13 per kilowatt year,—or say in the ratio of 1: 1.8. Although there may be advantages in the use of an electrical blast furnace instead of an ordinary blast furnace, these will not be taken into consideration in this connection, and we will assume that the cost of electrodes and of the electricity on the one hand, and its equivalent in heating value of coke on the other, would be the same; in other words, that the cost of electrodes and electricity would balance the cost of the amount of coke that would be necessary to do the same amount of work. We will not take into consideration the saving in blowing by reason of not having to furnish air for burning the coke.

"The average consumption of electrodes per ton of iron produced at Trollhättan, Sweden, is about 10 lb. Assuming 1 ton of iron is equivalent to 2 tons of charge, the consumption of electrodes per ton of charge smelted in a furnace, such as the one we have been discussing, would not be over 5 lb. per ton. If we assume that the electrodes do not cost over 6 cents per pound, which allows a fair margin for transportation over their cost at the factory, the cost of electrodes would be about 30 cents per ton of charge smelted. The consumption would be probably considerably less than 5 lb. per ton, as was shown in an experiment performed by the writers in which air was blown into the furnace, and in which the electric consumption was only 2 lb. per ton of ore treated. Such being the case, if our combined electrical energy and electrode cost is not to exceed our equivalent coke cost, with coke at $9 a ton, the cost of electrical energy would be $9 \times 1.8 = \$16.20$ a kilowatt year, minus the cost of electrodes, or about $16.00. Compared on the basis of calories, if the coke contains 80.24 per cent fixed carbon, it would contain 1604.8 lb. of carbon per ton of 2000 lb., and this carbon would, if completely burned to CO_2 give a calorific value of about 13,000,000 calories. As one kilowatt year of energy is equal to 16,616,000 cal., it would require something like 1.2 tons of coke, containing 80.24 per cent fixed carbon, to equal in calorific value the kilowatt year, but it is to be remembered that this assumes that all the carbon of the coke is completely burned to carbon dioxide, which rarely happens in practice, and so, as has been determined from actual practice, it requires from 1.5 to 1.8 tons of coke to yield the same number of calories as may be obtained from 1 kilowatt year of electrical energy."

THE USE OF THE ELECTRIC BLAST FURNACE IN THE SMELTING OF PYRITIC ORES

"As previously stated, in all true pyritic smelting, fuel (usually coke) in varying amounts is added to the charge. We will assume for the sake of argument, that the ore being treated is entirely suited to pyritic smelting, and that it is not necessary to use more than 0.5 per cent of the weight of the charge in coke. Could such an ore be treated pure as well in an electric blast furnace as in an ordinary blast furnace, assuming that the costs would be the same in each case. As what we have had to say in regard to the construction of the furnace and the costs of applying the process is as applicable to true pyritic as it is to semi-pyritic smelting, we will only take up the subject from the metallurgical and chemical standpoint.

"In the first place let us consider the nature of the atmosphere in a pyrite furnace. As the air enters the tuyeres, practically all of it enters into immediate combination, forming SO_2 and FeO. Although some oxygen escapes combination, it is not sufficient to support combustion within the furnace, as has been shown by Sticht and others. Now let us breifly consider the changes which take place in the charge as it descends from the charging door to the fusion zone.

"They are few, and may be briefly stated as follows:

(1) "The iron sulphide loses one-half of its sulphur as elemental sulphur, and becomes FeS.

(2) "Chalcopyrite. It loses about one-fourth of its sulphur and becomes practically a mixture of Cu_2S and FeS.

(3) "The silica is unchanged.

(4) "The coke is for the most part consumed in the upper zone of the furnace, but not by oxygen, as no free oxygen exists in that part of the shaft, but by the O from SO_2, that is, the carbon of the coke reduces the SO_2 giving S and CO, and this CO reacts with more SO_2, forming S and CO_2. Aside from these reactions, which are of no great importance in the chemistry of the process, the chief office of the coke as pointed out by Sticht,[1] 'Is apparently to heat up the sulphides and the quartz in preparation for their active oxidation deeper in the furnace.' If, therefore, coke was omitted from the charge we would not have this preparation of the sulphides and the quartz as stated by Sticht. Such being the case, how detrimental to the process would this lack of preparation be?

"In order to answer this question, it must be borne in mind that in true pyritic smelting, there is 'no heat to spare,' and that although, as pointed out by Sticht, the heat evolved by the combustion of the coke by the oxygen of sulphur dioxide is only one-third as much as it would be if the oxygen were furnished direct by the blast, and that it is given off at a point considerably higher than the focus of the furnace, it nevertheless 'assists in preparing the charge for the reactions which take place in the focus of the furnace, for with 1 per cent of coke in the charge enough heat is generated to supply one-third of the heat required to melt the entire pyrrhotite contents of the charge.' This extra heat 'doubtless furnishes just the necessary aid to bridge the operation over some critical point.'

"From this we see that the office of the coke then is solely for the purpose of furnishing heat. Inasmuch as the nature of the charge in true pyritic smelting is such as to demand the oxidation and removal of a large excess of iron of silicate, and as it is necessary to cause the silica present to unite with as much FeO as possible (due to the fact that

[1] *Principles of Copper Smelting*, by E. D. Peters, p. 227 (1907).

generally silica-bearing materials have to be added for slag-forming purposes, and often these are barren and hard to obtain), and as the saturation point of silica with iron oxide is greater the higher the temperature attainable in the pyritic furnace, it is quite necessary that the reactions in the zone of the furnace proceed as energetically as possible, in order to meet the requirements stated above, and to provide enough heat to keep the process going.

"If, therefore, pyritic smelting be attempted in an electric blast furnace, it would be necessary to supply the heat from the electric current in such a manner as to insure there being enough heat at the focus to carry on the reactions at that point in the same manner as if coke had been added to the charge. The writers are of the opinion that it would be possible and perfectly feasible to do this in the furnace of the type shown in Fig. 3. Moreover, that the use of the electric furnace would permit of a more uniform operation of the furnace than is now possible, for, with the electric current, it would be an easy matter to quickly increase or decrease the heat required for the successful operation of the furnace. In this way we could maintain the 'degree of concentration at a constant,' which, as pointed out by Sticht, is very important, but which is extremely difficult to do. Also, we could avoid the troubles arising from the 'constant fluctuations' which are bound to occur, and which at present demand unbroken attention and frequent change in the charge, especially in its proportion of silica 'due to the fact that in true pyritic smelting there is no heat to spare,' whereas, with the electric blast furnace, it would be possible, as just stated, to increase or decrease the heat as necessary in order to meet these fluctuations.

SUMMARY

"Summarized then we may say that the electric smelting of copper ores is nothing more than the substitution of electric heat for the heat derived from the combustion of carbon. Inasmuch as the carbon which is used either in the reverberatory furnace or in the blast furnace plays no important part in the necessary reactions which take place in these furnaces, there is no reason, metallurgically, why electric heat may not be substituted for the heat derived from the combustion of carbon. In fact, as we have tried to point out, in some cases the reactions would take place to better advantage in the neutral atmosphere of the electric furnace than in the reducing or partially reducing atmosphere of the combustion furnace. Therefore, as to whether the electric furnace would be used for the smelting of copper ores would largely depend, so far as we are able to make out at the present time, upon the relative cost of coke and electric power.

"As the use of the electric furnace is not advocated as a competitor of the combustion furnace, but as a substitute for it in those localities where it is not advisable because of the high cost of fuel, we see no reason why the electric furnace may not be developed as a substitute for the combustion furnace, where the conditions are such as to warrant its use, especially in the treatment of copper-bearing ores. In this connection it is to be remembered that the reason why the electric furnace was developed in the iron industry, for the reduction of iron from its ores was due to necessity. As a matter of fact the field for the electric furnace in the reduction of iron from its ores is a limited one. Perhaps the name is true as regards the possible application of the electric furnace to the treatment of copper ores, but, judging from the comparative costs as shown in the preceding pages, it would seem that the chances in favor of the electric furnace for the treatment of copper ores are greater than those for the treatment of iron ores, because there is not so great a difference in the cost of coke and electric power in copper mining districts as in iron smelting centers. Also the cost of electric power is constantly becoming less, due to improvements in gas engines and steam turbines, so that, in districts where water power is not plentiful but cheap fuels unsuited to coking purposes are available, it may be found more advantageous to use electric heat than the heat derived from the combustion of coke.

"It is sincerely hoped by the writers, that although it is possible at this time to present but few facts, that the comparative study herewith presented may serve to stimulate interest in the subject of the electric smelting of copper ores, and to cause others to attempt a further investigation of it. As a result of their investigations, the writers are convinced that experimental work on a larger scale should lead to the development of an electric blast furnace, which in some cases could be used to better advantage for smelting sulphide ores of copper than the combustion blast furnace."

Sunday—Helena

On Sunday, August 17th, luncheon was taken at noon in Helena and in the afternoon the East Helena lead-silver smelter of the American Smelting & Refining Company was visited.

Monday—Butte

On Monday in Butte the attendance had increased to 150. The morning session in the Auditorium was opened by an address of welcome by C. F. Kelly, Vice-President of the Anaconda Copper Mining Company.

Papers were then presented by R. H. Sales, D. C. Bard and M. H. Gidel, B. H. Dunshee, N. P. Brady, J. Gillie.

Geological and Mining Papers

The very elaborate paper of 100 pages, by **Reno H. Sales,** Butte, Mont., dincussed the geology of the "ore deposits at Butte, Montana." It is impossible to do justice to the paper in an abstract which would be necessarily short. In the discussion Grattan did not agree with the main conclusion as to the origin of the chalcocite Butte ore.

The paper by **D. C. Bard** and **M. H. Gidel,** Butte, Mont., discussed "mineral associations at Butte, Mont."

The paper by **B. H. Dunshee,** Butte, Mont., dealt with "timbering in the Butte mines."

The paper by **Norman T. Brady,** Butte, Mont., discussed "shaft-sinking methods of Butte."

The paper by **John Gillie** dealt with the "use of electricity in the Butte district."

Excursions—Leaching Low-Grade Copper Ores

In the afternoon of Monday eight different parties were formed, which visited the electrically-driven air-compressor plants and air hoists at the High Ore Mine, the underground workings and pumps at the Original, Leonard, North Butte, and Tramway Mines, the underground workings and the zinc mill of the Butte & Superior Company, the mine and the copper-leaching plants of the Bullwhacker Copper Company and of the Butte and Duluth Mining Company.

Most interesting was probably the inspection of the Butte & Duluth Mining Company's leaching process for low-grade ores with subsequent electrolysis with insoluble anodes. The present output is over one ton of copper daily from a two per cent ore. This is to be increased to ten tons daily.

Those who saw the process were greatly impressed by its potentialities.

It seems not unlikely that it may revolutionize the treatment of carbonate and silicate low-grade ores.

Basic-Lined Converter

The first paper presented in the technical session on the evening of Monday was the paper by **Mr. E. P. Mathewson,** of Anaconda, Mont., on the "Development of the Basic-Lined Converter for Copper Mattes."

The author gave a historical sketch of early unsuccessful experiments and outlined the later successful development of the basic-lined copper converter. It summarizes the main points to be observed for success as follows: "Not to exceed a temperature of 2100 deg. F.; not to have tuyeres smaller than 1.25 in. (1.5 in. is the preferred size); to drive in punch rods the full size of the tuyere opening, immediately after pouring copper; to maintain in the converter as large a mass

of matte and slag as possible to prevent sudden changes in temperature and overheating of the lining; to employ slag containing preferably about 25 per cent. of silica."

The following test was made at Anaconda with a view to finding out whether the cutting action of converter slag on a magnesia brick lining bore any relation to the silica content of the slag.

Thirteen slags were selected from the daily samples sent to the laboratory. These varied in silica from 22.4 per cent. to 37.8 per cent. and were carefully analyzed for MgO. The results given in a table in the paper show no relation between silica and magnesia content.

Mr. Mathewson concludes: "To Smith and Pierce belongs the credit of taking a long-discarded idea and developing it into a successful process. The great advantages of the process are: decreased cost of lining; the ability to use large units in converting, with consequent economies in labor, power and repairs; neatness and cleanliness of plant, abolishing the danger, from dust, to the health of the lining crew."

The paper was discussed by Dr. Crossdale and Dr. Richards.

Round Tables

A paper by Mr. **Theodore Simons,** of Butte, Mont., discussed "the evolution of the round table for the treatment of metalliferous slimes."

The author emphasizes the great importance which has been attached in recent years to the treatment of very fine sands and slimes. "For the profitable treatment of the very finest slimes, round tables have as a last resort proven more satisfactory and economical than other machines."

After discussing the general principles of "film sizing" the author outlines the evolution of the round table, beginning with the primitive dressing plant described in Agricola's de Re Metallica. From rectangular buddles he passes over to round tables and distinguishes round tables of the intermittent type (convex and concave types), and round tables of the continuous type.

Among those of the latter type the following are described in some detail: 1. Revolving tables with stationary fixtures (Evans). 2. Stationary tables with revolving fixtures (Linkenbach). 3. Percussion round tables (Bartsch). 4. Revolving round tables with adjustable inclination of the deck (Demuth).

While in all these tables the generatrix of the deck surface is a straight line, tests are being made at present at Great Falls with round tables, having a deck the generatrix of which is the arc of a circle, whose chord has an inclination from the horizon varying with the fineness of the pulp to be treated.

Tuesday (Anaconda)

In the morning of Tuesday the members and guests of the Institute were carried by a special train to Anaconda, where the forenoon was spent in an inspection of the Washoe smelter.

In the afternoon a technical session was held at the Margaret Theater, opened by an address of welcome by Mr. E. P. Mathewson, the reply being made by Dr. J. W. Richards. Papers were then presented by Ralph Hayden, Albert E. Wiggin, Robert Ammon, E. S. Bardwell, Edgar M. Dunn, James O. Elton, Robert H. Richards, and E. G. Woodruff.

Slime Concentration at Anaconda

The paper by **Ralph Hayden,** Anaconda, Mont., first discussed briefly what "Anaconda slime" is, its source and its constitution. "The problem of wet concentration of the Anaconda slimes may be divided into two parts, viz.: First, to prepare the slime for use on some concentrating machine by thickening it to the proper density, and at the same time recover water clean enough for reuse in the mill; and second, to recover the mineral values from the thickened pulp in a concentrate which can be smelted."

The author then passed over the experimental slime tank at Anaconda and described at some length the slime thickeners in use (slime ponds; tanks; comparative efficiency of open and baffle tanks; fallow tanks; Kuchs-Laist centrifugal separator; Garred filter; Dorr continuous thickener) and concentrators (round table; Peck centrifugal concentrator; competitive tests; twenty-deck round table; slime feed distributor). Figures are given of the results of tests, often of several days' duration.

The final conclusions are as follows:

"It is evident that large settling areas are required to thicken the slime where free settling is to be used. If filters or centrifugal machines are resorted to, then space can be economized, but increased maintenance costs and operation costs, together with complex machines, enter into the problem.

"The Door tank has nearly the same capacity per square foot of settling area as the smaller units; namely, 0.20 to 0.25 gal. per square foot per minute; but it has the advantage of large units, say 50-ft. tanks, which can be built 3 ft. deep and three in a stand; and consequently requires few discharge plugs, and these of good size. The filter is a very compact and efficient machine, but has the disadvantage of operating in cycles as well as having high maintenance costs.

"The Kuchs-Laist separator is very compact and is continuous in operation, but requires power, and also a satisfactory discharge plug to give constant density. Its product is freer of colloids than that of the other thickeners, which allows concentrators to make a greater net recovery from it. If a uniform spigot density could be maintained this machine would be very satisfactory.

"The round table has proved very efficient for handling slime, it being simple in construction and operation, and a number of decks may be placed one above the other, thus economizing on floor space, where the mill height is available.

"The Peck machine has a good principle, namely, the utilization of the effect of centrifugal force, which allows it to handle a good tonnage in a small space. It makes a good recovery on the slime settled in the ponds.

"The best net recovery of copper in the original slime is from 50.0 to 52.0 per cent. except where the Kuchs-Laist machine is used. The slime contains about 20 per cent. of collodial copper, which is not recoverable by wet concentration; and there are middling grains to be considered as well. Thus the net recovery of recoverable copper is probably 65 per cent., which is very good for wet concentration on this class of material."

Other Important Papers On Concentration and On Smelter Fumes

More papers followed on concentration practice in Anaconda and Great Falls. In view of the importance of these papers and on account of the limitations of space in this issue, an analysis of these papers is reserved for the next issue, and only the titles of the papers presented will be given here:

"The Great Falls System of Concentration." By **Albert E. Wiggin** (with chapters on the early history of the Boston & Montana mill at Great Falls from 1891 to 1905; the replacement of the Evans' jig by the Hancock jig; the Richards' pulsator classifier system; the Woodbury classifier and jig system, the Great Falls' System, and a comparison of the Great Falls' and Evans' systems at Anaconda).

"The Anaconda Classifier." By **Robert Ammon** (giving a sketch of this hindered-settling classifier and an account of the actual results obtained in practice with the classifier working on the Butte copper ores at Great Falls and Anaconda).

"Application of Hindered-Settling to Hydraulic Classifiers." By **Earl S. Bardwell** (giving data from practice on the question of proper ratio between the area of sorting column and the area of constriction).

"Ore Dressing Improvements." By **Robert H. Richards** (read by Mr. Bradley Stoughton).

The following two papers dealt with the problem of smelter fumes, and abstracts of them must also be reserved for our next issue:

"Determination of Gases in Smelter Flues, and Notes on the Determination of Dust Losses at the Washoe Reduction Works, Anaconda, Mont." By **Edgar M. Dunn.**

"Arsenic Trioxide from Flue Dust." By **James O. Elton.**

A paper by **G. E. Woodruff** on "Typographic Maps for

Mining Engineers" was also read at this afternoon session.

In the evening the members and guests of the Institute were taken by train to Butte, where in the Auditorium moving pictures showing the Montana power plants, etc., were greatly enjoyed. Not less enjoyable was the dance "indulged in by those who wished" at the Columbia Gardens later on.

Wednesday (Butte)

In the Wednesday morning session, during which Dr. J. W. Richards was in the chair, papers were presented by Messrs. Johnson, Saunders, Roberts, Billingsley, Guinniss, Herewith, and van Barnevelt.

Reducibility of Metallic Oxides As Affected By Treatment

The paper on this subject, by **Woolsey McA. Johnson,** of Hartford, Conn., emphasizes the fact that the ease of reduction of metallic oxides depends largely on the way they have been prepared.

The temperature at which zinc oxide evolves zinc according to the equation

$$ZnO + C = Zn + CO$$

depends on the nature of the two reacting bodies.

In general, all equations reacting in the liquid or the gaseous phase have well-defined constant physical conditions which govern the equilibrium point and reaction velocities. But in a reaction where we have one or more solid phases, the temperature conditions requisite for the unbalancing of the stable system are found to vary. "These variations are due to the fact that in a solid there is a very complex molecule, and it is not a question solely of the tearing away of two interlocked atoms, but also of the destruction of a complex molecule containing hundreds, or possibly thousands, of smaller molecules.

"In other words, there are two forces to be overcome: A certain 'chemical' force, F_c and a certain 'physical' force, F_p. Theory indicates that F_c is constant and F_p is variable. F_p depends on the physical configuration of the small molecules of the metallic oxide."

The author described his "reaction tester" devised for such work, and gave some of the results obtained with it.

C.P. zinc oxide made by the wet method and soft 600 deg. C. charcoal gave a reaction temperature of 1022 deg. C.; 48-hour Indian territory coke gave with the same substance a temperature of 1029.5 deg. C.; with Acheson graphite turnings the reduction temperature was 1085 deg. C.

The same substance, C.P. zinc oxide, was treated by heating at definite temperatures for 12 hours with soft coke, and its reaction temperature was determined. The three values of C.P. wet-method zinc oxide are as follows:

	Reduction Temperature
No previous treatment	1029.5° C.
1100° C. treatment	1048° C.
1300° C. treatment	1061° C.

From these and other results given in the paper the author deduced the general law: *The higher either of the reacting solid bodies is previously heated the higher is the reduction temperature.*

The author discussed briefly the rationale of this law, and then continued:

"Taking up the question of the changes in the nature of the zinc oxide as affecting reduction temperature, it might be well to consider zinc ferrite, zinc silicate, and zinc aluminate. All these have been made by the writer by heating Fe_2O_3, SiO_2, and Al_2O_3 mixed with molecular equivalents of zinc oxide. They are insoluble in ammonium chloride and dilute acids, and, if strongly heated, insoluble in concentrated acids. They are also inert to reduction by carbon and have high reduction temperatures.

"We can, however, derive a useful lesson from these compounds. For, *if zinc oxide can combine with metallic oxides as aluminum oxide and iron oxides, why can it not combine with another metallic oxide: to wit, zinc oxide?* Accordingly, we can say that, when zinc ores are heated very strongly, 'zinc zincate' is formed, and that the energy of this compound must be supplied to break it up before the reaction can proceed.

"The writer has found that these experimental facts agreed with the practical experience of himself and many others in the zinc business; for instance, it is a fact that 'spiegel oxide,' which is formed in the downcomers and in the hot-blast stoves of the New Jersey Zinc Co.'s spiegeleisen furnaces working on residues from the 'oxide' furnaces, and which is thus made at a low temperature, requires little heat in 'retorting,' whereas 'refuse oxide,' made in the Wetherill furnaces of the same company at a temperature vastly higher, requires a much higher temperature to 'retort.' 'Dead' coal, or coal from the strip pits of western Missouri, has been known to reduce zinc oxide much more easily than Arkansas semi-anthracite, as the tests show. It is natural to expect that the weathering action makes the carbon more active."

After discussing the initial temperature of reduction, the author deals with the rate of reduction. His concluding remarks are:

"Every item of experimental evidence and theoretical speculation indicates that the reducibility of zinc oxide, or any other metallic oxide, depends on the way in which the reducing agent and the material to be reduced are heat treated. Practically we find this to be the case in retorting and also in smelting in the writer's electric furnace. In the latter the reactions proceed in an entirely different manner, for the alternating current is a catalyzer and the slag of the smelting zone introduces new features besides, and only enough carbon is added to reduce the zinc oxide.

"Since researches prove that such practical changes as roasting at as low a temperature as will give a 'dead' roast increase the ease of retorting to such an extent that it is directly and distinctly sensible to the workman, it is but logical to reason conversely and to attribute to the practical judgment of the workman a substantial degree of accuracy. Now, any practical zinc man knows that ores from one mine will 'retort' easily and ores from another will 'retort' slowly. So it is hardly to be doubted that an investigation of the reduction temperature of zinc blende roasted under standard conditions, and with a standard reducing agent, in a Johnson 'reaction tester' would determine the geological history of such mines. The writer knows little about geology, but he is sure that blende deposited primarily would give when roasted a reduction temperature 10 deg. C., and possibly 20 deg. C., higher than a blende deposited in the same mine secondarily and tested under same conditions."

Blast-Furnace Jackets

A paper by **Robert P. Roberts,** Great Falls, Mont., on the "Thermal Effect of Blast-Furnace Jackets," gives an account of quite an elaborate test of the water consumption in these jackets and the heat thereby carried away. An analysis of these results must be reserved for a future issue.

The following figures were also presented at this session: Rock Drill Economics. By **William L. Saunders.** The Southern Cross Mine, Georgetown, Mont. By **Paul Billingsley.** Occurrence and Manufacture of Refractories in Montana. By **Guinniss.** Mining and Metallurgy at the Panama Exposition. By **C. E. van Barnevelt.**

The afternoon of Wednesday was again spent in various excursions in the neighborhood of Butte.

Concluding Technical Session

The concluding technical session was held on the evening of Wednesday, when papers by Febles, Burns, Corwin and Rogers, Krejci, Bardwell, Nordberg were presented, abstracts of which are herewith given.

Precipitation of Copper from Mine Waters

A paper by **John C. Febles** on "the precipitation of copper from the mine waters of the Butte district" gives first, a historical sketch.

The principal reaction which occurs is the precipitation of metallic copper at the expense of metallic iron, is

$$Fe + CuSO_4 = FeSO_4 + Cu.$$

The resulting ferrous sulphate becomes oxidized by constant

aeration in the flumes and towers to ferric sulphate, which, being subjected to still further oxidation, is thrown out of the solution, both as basic ferric sulphate and as ferric hydrate, with the liberation of free acid, which reacts upon either copper or iron salts, again forming the corresponding sulphates.

The quality as well as the quantity of the cement copper obtained in a given time depends upon a number of conditions, which may be briefly enumerated as follows:

1. The amount of copper in solution.
2. The temperature of the solution.
3. The freedom of the solution from salts other than copper.
4. The rate of flow.
5. The kind of precipitating apparatus used. (Flumes are superior to towers in almost every way.)
6. The kind of iron used. (Almost any merchant iron or mild steel is suitable; old rails, merchant bar, and pipe of small diameter seems to be the best.)
7. The size and shape of the iron, and the way it is placed.
8. The proper handling and cleaning of the iron.

The author then describes the High Ore plant and its equipment. It produces approximately 2,200,000 lb. of pure copper annually, which amounts to the shipping of about ten cars of crude precipitate each month.

The iron consumed in the process weighs from 1.1 to 2 times as much as the copper produced. The lower the value in copper, or the higher the ferric sulphate or free acid, the greater the consumption of iron. Good, clean, compact iron will precipitate more copper than tin cans and similar light iron. At the Silver Bow plant, where a large percentage of railroad rails and similar desirable iron is used, 1700 lb. of copper recovered per ton of iron used is claimed. A fair estimate for normal conditions would be 1 lb. of copper recovered for each 1.5 lb. of iron consumed.

The author then discusses the other products precipitated. The presence of the ferric hydrate, or "ocher" in the precipitate is objectionable, and is to be avoided as far as possible. There does not seem, however, to be any satisfactory way to control its precipitation. The formation of this precipitate is due to oxidation of iron in solution, probably by aeration, which results in its being thrown down both as ferric hydrate and as basic ferric sulphate.

The mine waters of Butte also contain small amounts of arsenic, part of which is precipitated with the copper, although not proportionally with the copper.

An average of cost figures, extending over one year's operation of the Leonard plant, is as follows:

	Per lb. of Copper Produced
Labor	$0.0352
Supplies and iron	0.0140
Sundries	0.0002
	$0.0494

The above figures are upon the basis of the net pounds of copper contained in the precipitate shipped and paid for.

The author concludes with a brief description of the methods of protection of the pumps and water columns from the acid water.

The water ends of all pumps are constructed, as far as possible, of a phosphor-bronze alloy analyzing approximately 85 per cent of copper and 15 per cent of tin. Experience has shown that this alloy will stand the corrosive action better than any material as yet tried.

Plungers and valves made of this bronze are not altogether proof against the action of the water, however, having to be replaced from time to time. The corroded parts are, upon removal, sent to the machine department for repairs, a large supply of interchangeable parts being kept on hand.

The columns are constructed of cast-iron pipe, lined with either lead or wood. The lead-lined pipes are flanged, the lead lining flaring, so that when bolted together, with a rubber gasket between, a water-tight joint is insured, and the iron casing is completely protected from the water flowing through. The wood linings are made of narrow strips, the last strip being driven firmly home, forming a key, which holds the lining in place. The whole inside is then painted with acid-proof paint, and all joints are carefully calked and painted to prevent leakage. The exterior of all pump columns is protected with acid-proof paint.

In the discussion W. T. Burns gave a brief account of an attempt made at the Great Falls electrolytic plant to recover the copper from the mine water by direct electrolysis. His conclusions are that the recovery of the copper from the mine water by direct electrolysis does not appear to be possible from a commercial standpoint, chiefly on account of the low electric conductivity of the mine water, resulting in a high power consumption.

Electrolytic Refining of Copper Precipitate Anodes

A paper by **W. T. Burns** gives "notes on the electrolytic refining of copper precipitate anodes."

Attempts were made in 1908, at the Great Falls Works, to produce ingots direct from the Butte precipitate by smelting the material in a reverberatory refining furnace. The ingots produced in this manner average 99.4 per cent of copper and 0.40 per cent of arsenic and antimony.

The copper content being considered too low, it was decided to cast the copper into anodes and to treat the anodes in the electrolytic plant.

Details are given of the results of the treatment and the conclusions are summed up as follows:

Cathodes of wire-bar grade can be economically produced from copper precipitate anodes when treated at a current density of 17 to 18 amp. per square foot.

Cathodes, from which cake and ingots averaging 99.85 per cent of copper can be made, can be economically produced from copper precipitate anodes at a current density of 33 to 35 amp. per square foot.

The removal of the arsenic from the electrolyte is the principal item of electrolytic refining cost that is in excess of the cost to refine ordinary converter anodes.

It would then appear that if anodes can be produced as cheaply in the refining furnace as the precipitate can be treated in the smelter, there is little choice between the two methods, as the copper must eventually pass through the electrolytic refinery.

MacDougall Roasters

A paper by **Frank R. Corwin** and **Selden G. Rogers** on "increasing the efficiency of MacDougall roasters at the Great Falls Smelter of the Anaconda Copper Mining Company," points out that since the first installation of MacDougall roasters at the Great Falls smelter of the Anaconda Copper Mining Company, the capacity of the furnaces has been more than doubled. During the first nine or ten years of the operation of the MacDougall department the tonnage treated by the furnaces remained practically unchanged. Experimental work carried out in 1906, however, showed that increasing the draft of the furnaces would bring about an increase in capacity. In 1909 the MacDougall department was connected to the new smelter stack and flue system, and the stronger draft caused by this change increased the capacity of the furnaces. The change in draft conditions made it necessary to operate differently, and systematic experimental work was started to adapt the furnaces to the new conditions.

This experimental work extended over a period of several years, and resulted in very largely increasing the capacity of the MacDougall furnaces, decreasing the percentage of flue dust, and also brought about other changes and improvements tending to raise the efficiency of the MacDougall department.

These changes, of course, resulted in reducing operating expenses. The improvement was brought about without building any new MacDougalls or enlarging the old ones, and was the result of the thorough study of the MacDougalls and numerous tests.

The paper gives a detailed account of the experimental work by which the increase in efficiency was brought about. An

analysis of this work must be reserved for a future occasion.

The following papers were finally presented:

The Metaline Plant of the Inland Portland Cement Company, Metaline Falls, Wash. By **Milo W. Krejci.**

Notes on the Metallography of Refined Copper. By **Earl S. Bardwell.**

The Compressed Air System of the Anaconda Mining Company, Butte, Mont. By **Bruno Nordberg.**

The program for Thursday included an all-day trip to the Southern Cross Mine and a banquet at the Silver Bow Club in the evening.

A magnificently illustrated pamphlet prepared by the local committees for this convention deserves special mention. The Montana section of the Institute was officially established by the directors in their session of Wednesday. The whole convention was greatly enjoyable, highly important, and most successful.

Synopsis of Recent Chemical and Metallurgical Literature

Iron and Steel

Electrolytic Prevention of Corrosion.—A preliminary report has been issued by the Bureau of Mines, *Technical Paper* No. 15, giving results of experimentation in electrolytic methods of preventing corrosion of iron and steel. The Bureau has undertaken a thorough investigation of the problem, and this report is merely an account of experiments made looking toward the development of a satisfactory method.

Experiments begun in 1911 by Mr. **F. M. Stanton**, then chemist at the experiment station in Pittsburgh, consisted in exposing steel plates in separate vessels in $\frac{N}{5}-H_2SO_4$. In one vessel the steel plate was connected with the negative pole of a storage battery, the positive pole being connected with a carbon rod immersed in the solution near the steel plate. In another vessel the steel plate was left unprotected by any current. After 24 hours' exposure the unprotected plate had lost 1.81 g. weight, and the protected plate only 0.0012 g. In tests with plates submerged in the Monongahela river, the current density was less than in the laboratory experiment, and the protection against corrosion was less complete. The current was furnished by a 2-volt accumulator. The loss in 15 days was 0.7 g. for the protected plate and 10.9 g. for the unprotected. Mr. Stanton's experiments were not completed, and further work was undertaken by Messrs. **A. E. Hall, J. K. Clement** and **L. V. Walker.**

Their first experiments were not satisfactory, but indicated that a definite current density must be maintained in order to reduce corrosion to a minimum. Subsequent experiments showed that a current density of 0.4 milliampere per sq. in. would prevent 93 per cent to 94 per cent of the corrosive action of $\frac{N}{10}-H_2SO_4$ when the solution is not stirred.

In experiments with stirred electrolyte, plates having a current density of 0.4 milliampere per sq. in. or less continued to lose weight during the entire period of experiment. Plates protected by current densities greater than 0.4 milliampere showed a marked initial corrosion, but thereafter no further loss in weight.

To determine the effect of oxygen in the electrolyte, experiments were made in which the solution was saturated with oxygen, and the results showed that corrosion was rapidly accelerated by increasing the amount of that element in the electrolyte. Corrosion increased also with the increase in the rate of stirring the electrolyte. Experiments to show the effect of acid concentration show that although a considerably greater current density would be required, the method would be effectual even in $\frac{N}{10}-H_2SO_4$. With $\frac{N}{100}$ and $\frac{N}{1000}$ acid the results were practically identical, showing that between these limits of acid concentration there is no change in the rate of corrosion.

Gold and Silver

Vacuum Filtration in New Zealand.—An elaborate article on the plant and operating results with the Moore process at the Victoria mill of the Waihi Gold Mining Company, New Zealand, is published by **Mr. William Macdonald,** in the May, 1913, *Journal* of the Chemical, Metallurgical and Mining Society of South Africa. Filter presses were formerly used at this mill, but on account of their cumbersome nature and high cost of operation they were displaced in 1908 by the Moore filter.

A brief statement of the metallurgical process at the Victoria mill follows: The ore is crushed in water, amalgamated, and separated into sand and slime products, the former being leached. The slime is thickened by settlement, dewatered by a Moore filter, mixed with cyanide solution, agitated and again filtered by the Moore process. The slime is very finely divided, 91.33 per cent passing a 200-mesh screen. It contains about 1 per cent pyrite.

A complete cycle of filtration occupies 144 minutes, distributed as follows: Loading, 50 minutes; washing, 60; discharging, 24; transfer of baskets, 10. With a partial vacuum rising to 24 in., and a pulp of 1.15:1 consistency, a 1½-in. cake is formed in about 50 minutes. The routine work of the plant proceeds smoothly when the pulp is of the proper consistency, but with a thin pulp is often the cause of trouble. Thin pulp may be due to several causes, but probably is accounted for by poor settling in the collecting tanks.

Filter cloths have a life of 12 to 15 months. It is customary to place new cloths on the cyanide filter for three months, then remove and treat with a 5 per cent hot solution hydrochloric acid, and, after washing and drying, use again on the cyanide filter for three months. After the second removal and acid treatment the cloths are used on the dewatering filter for six to nine months.

Daily assay is made of discharged cakes. A sample is taken from every other leaf of each basket throughout the 24 hours. After cutting to about 20 lb., the slime is wrapped in clean filter cloth and squeezed between plates by screwing up bolts. The solution thus obtained is assayed, using 20,000 gr. The squeezed pulp is then broken up and washed by decantation, and finally assayed in triplicate, taking 1000 gr. for each charge. The amount of soluble gold left in the discharged cakes is less than 1 gr. per ton of dry slime. The cake is discharged with 30 per cent moisture. Filtering capacity per sq. ft. filtering area is about 10 lb. On a 144-min. cycle the maximum number of charges per day per basket is 10.

For the year 1911 the average assay of cake as discharged was 0.598 dwt. gold and 0.768 oz. silver per ton; average assay of washed cake was 0.55 dwt. gold and 0.746 oz. silver per ton. The extraction was 89.8 per cent of the gold and 71.5 per cent of the silver. The efficiency of displacement is given as approximately 99 per cent. The cost per ton of dry slime treated during 1911 was as follows: Labor, 5.54 cents; power, 2.88; supplies, 1.14; assaying, 0.4; total, 9.96 cents. The average cost of power from producer-gas plant was 15.974 cents per hp-day.

In conclusion it should be remembered that the process in use at this mill differs from the most modern practice in crushing in water, thus necessitating dewatering before cyaniding, and in relying on settling tanks for producing thickened slime.

Zinc and Lead

Bisulphite Process.—The British Metals Extraction Co., Ltd., is using the bisulphite process on zinc-lead ores at Llansamlet, Wales. According to a report made at the last annual meeting of the company the process is conducted as follows: The ore is pulverized and roasted slowly in mechanical furnaces. The sulphurous gases are conducted from the furnace to towers in which the extraction is made. The roasted ore is mixed with water to form a thin pulp which is

allowed to descend through the towers, meeting and mixing with the ascending gases. The interior construction of the towers is such as to cause intimate and effective contact of the gas and pulp. The solution of zinc bisulphite thus formed is separated and heated in a special apparatus, forming a precipitate which is calcined, yielding zinc oxide. The latter is then smelted, producing spelter assaying 99.85 per cent. zinc. No difficulty is experienced in the smelting operation. The extraction of zinc is reported to be 84 per cent to 89 per cent, depending on the efficiency of the roast. The lead residue contains about 8½ per cent zinc, and is smelted.

The Development of a New Optical Pyrometer

By C. G. Fisher

Since the introduction of Wedgewood's pyroscope in 1782, the matter of high-temperature measurements has received the attention of some of the most competent scientists of their times. Many valuable improvements in all forms of pyrometers have been made, but only those relating to the optical pyrometer are referred to in this article.

FIG. 1.—WANNER PYROMETER

The goal toward which all manufacturers of pyrometers have been striving was to produce an accurate instrument which would be suitable for use in the industrial plant. This requires an instrument which is not cumbersome, not too delicate to stand the ordinary usage under the conditions as they exist, an instrument which will not deteriorate rapidly when being subjected to the necessary operations, one which also could be satisfactorily handled by the workman of average intelligence. Each of the improvements in the optical pyrometer has been a step in this direction until the very latest improvement, the first announcement of which is this article, brings it into the position where it fulfills the requirements of a practical temperature-measuring instrument for use in the manufacturing plant.

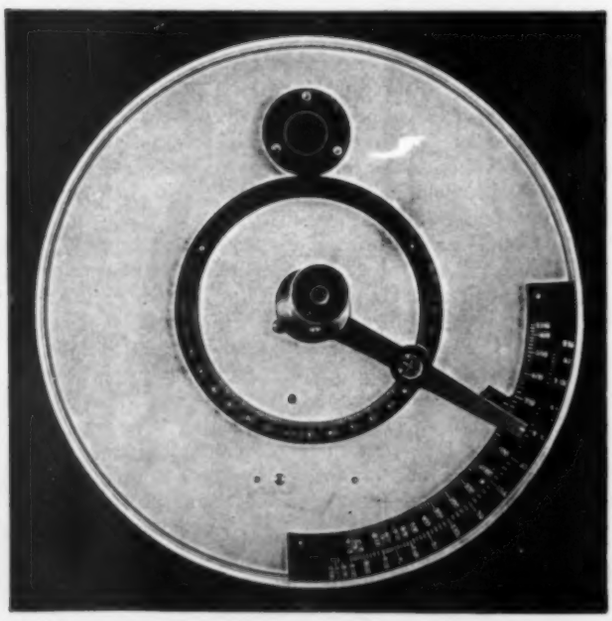


FIG. 2.—DIRECT-READING TEMPERATURE SCALE

It is interesting to trace the successive improvements in the optical pyrometer, each one of which corrects a feature which interfered with the practical application of the instrument. The general type of instrument which has made the greatest strides is the one based on Wien's law, utilizing the photometric measurement of light radiation of a given wave length of a definite portion of the visible spectrum. The Wanner and the Scimatco instruments are of this type and this is the type referred to here.

Wien's law, which expresses the physical principle upon which this type of pyrometer is designed, has been checked experimentally throughout the widest ranges of temperature and has received the most convincing verification.

In this instrument there are two adjacent fields of vision, one being illuminated by a standard incandescent lamp and the other by the object whose temperature is to be measured. This latter field must be adjusted to the same intensity as the standard field, and this is accomplished by turning the eye-piece.

It is generally understood that the optical pyrometer is not a low-temperature measuring instrument, its range beginning at about 650 deg. C. But as a high-temperature instrument it has no equal, and is now obtainable for taking accurate readings up to 7000 deg. C.

FIG. 3.—FURTHER IMPROVEMENTS IN CONSTRUCTION

The first optical pyrometer of this type which has been used to any great extent is the one designed by Professor Wanner in Germany (Fig. 1). This pyrometer has the good points of the instruments of this type, such as accuracy, high range of temperature, etc., but it also has objectionable features, the most conspicuous being a separate table to convert readings on the dial to degrees of temperature, incandescent lamp unprotected against breakage and easily thrown out of adjustment, and no protection against dust in the factory, which interferes with its proper working. The metal slit which admits the light may change in size, due to the expansion of the metal when subjected to the heat in front of the furnace. The instrument is confined to use where the object, whose temperature is desired, is under "black body" conditions.

It would be proper to mention here just what is meant by "black-body" conditions. It is known that different substances

FIG. 4.—SHOOK DIAL WITH OPEN SCALE

at the same temperatures when in the open, radiate different amounts of light, but should they be so situated that they are almost entirely surrounded by heated walls of uniform tem-

perature, as in a furnace, they then radiate the same amount of light, and are said to be under "black-body" conditions. The light radiation from a black body is due to temperature alone, and the physical principle on which the optical pyrometer was designed required that the object be under such conditions as make it a "black body."

The next important improvement was made by Dr. G. A. Shook in 1910, when he worked out the first direct-reading scale of temperature which was attached directly to the instrument, thus doing away with a separate reference table (Fig. 2). It might be well to mention that it has not been generally known that the credit for this improvement belongs to Dr. Shook, and such credit has never before been given him publicly.

This made the optical pyrometer the quickest reading instrument, as no time is required for the instrument to come up to temperature, and it requires only a few seconds to make the observations.

The question has been asked by many, "Is there not a great personal factor in matching the colors of the two fields?" having in mind, perhaps, the difficulty and accompanying complications of taking a piece of goods to town and buying a spool of silk to match. The matching of the two fields of vision is a great deal more simple, as the fields are adjacent, and as long as there is a difference in color, a line is visible between them, but just as soon as they match and one blends into the other the line immediately disappears. The operation, therefore, is simply turning the eye-piece of the instrument until the line disappears between the two fields, and as both are observed by one eye-piece and at the same time, any defect in sight is equalized.

Many experiments have been made with large numbers of different persons taking readings for the first time, and with the simple instruction to eliminate the line, the differences in readings were small, usually not more than 10 deg. between the highest and the lowest.

The next part of the pyrometer to be improved was the construction; the lamp and the wires leading to it were protected and the slit admitting the light was kept at a constant temperature so that its change of size due to expansion of the metal was eliminated. Dust also affected the size of the small opening and interfered materially with the operation of the instrument, and this was corrected by inserting a piece of special glass with known optical properties (Fig. 3). A much larger dial was designed for the instrument, allowing a larger temperature scale to be attached, which permitted more accurate readings, the dial also acting as a shield to protect the eye against outside influences. The whole instrument was now enclosed in a steel protecting tube which made it very substantial. This Scimatco pyrometer was adopted extensively by industrial works and used by workmen controlling temperatures in many different operations.

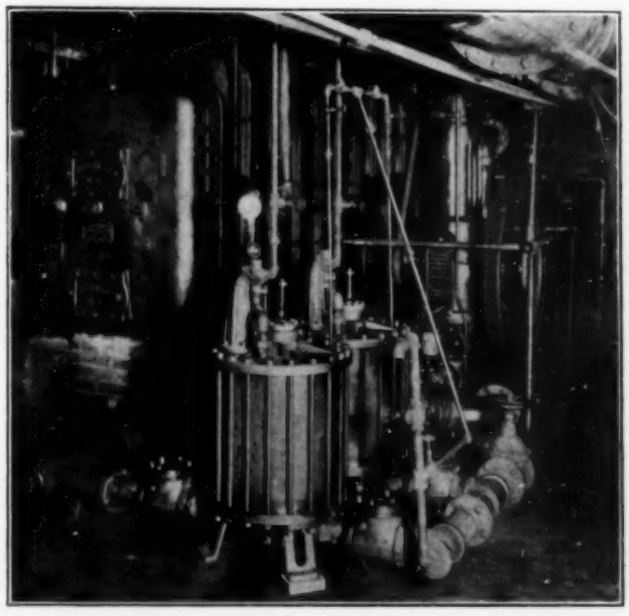
Dr. G. A. Shook of the University of Illinois has just perfected the latest improvement, a new optical pyrometer with a Shook open-temperature scale for the correct readings of temperatures of materials when not under "black-body" conditions. This new instrument is a universal one, having a scale for furnace temperatures in addition to the scale for open temperatures, and will, therefore, take the temperature of steel in the furnace, or while being poured from the ladle, the temperature of a billet, or a shape, while passing through the rolls, etc. This new pyrometer with the Shook open-scale has been based on the emissive power of light of different substances, calculated for the wave length, as used by the Scimatco instrument, and has been checked and found entirely correct by a number of scientists. The dial with the two scales is shown in Fig. 4. The first open-scale perfected was the one for iron and steel. Scales for the other metals came later, a different scale being required for different materials.

Dr. Shook has protected his new invention by applications for patents, all of which have been assigned to the Scientific Materials Co. of Pittsburgh, who have put the instrument on the market.

A Novel Application of Steam Traps for Evaporating Installations

For the last six months there has been an experiment tried out in the power plant of the Jacob Ruppert Brewery, New York, the successful results of which are interesting to all managers and engineers in factories where evaporators working under vaccum are used.

Part of the power plant of the brewery is run condensing, the condensing outfit consisting of two 1200-hp Wheeler condensers, each equipped with a single Edwards air pump and an efficient oil separator on the inlet. The condensation from these condensers is intended to be used in the ice plant for making distilled water ice. By examining the adjoining illustration it will be noted that the condensers are located close to the floor in the basement of the engine room, the actual distance from the floor being 30 in. Now, instead of the air pumps delivering water suitable for ice, an emulsion was delivered which consisted of the condensed water and the air and non-condensible gases churned together in the pump. The problem was to overcome this condition. This was done in the following manner:



STEAM TRAP

The air pump, at a slight expense and without disturbing the installation in any way, was turned into a dry vacuum pump, which removed all the air and non-condensible gases. A connection was made to the bottom of the condenser and led to two special vacuum traps designed and built by the Lytton Manufacturing Corporation, which removed the condensation from the condenser and delivered it to the ice plant. On account of the air and gases being taken out by the air pump the clear condensation flowed to the trap quietly and instead of being churned up was delivered in the same condition to the ice plant.

After the installation was properly installed the vacuum on the condenser was increased to 28½ in. The distance between the highest point to which the water rises in the trap, which is shown in the foreground of the illustration, and the bottom of the condenser is 6 in., and with this head two traps are able to handle a maximum amount of 400 lbs. per minute from the vacuum of 28½ in. This is done without any heating of the condensation or re-evaporation, because the time it takes to equalize from a 5-lb. pressure to a 28-in. vacuum when the trap is through discharging is only three seconds.

The construction of the trap is a vertical cylinder with inlet and outlet at the bottom and one of the regular Lytton steam and vent valves installed on top and outside where it is get-

at-able. On the inside of the cylinder is a floating piston of a neat design, which operates the steam valve.

These traps are automatic and adapt themselves entirely to the amount of condensation flowing to them. They need no oiling, no packing, no regulating and very little attention. The two shown in the cut have a 4-in. inlet and outlet with a maximum capacity of ten tons of condensation per hour each.

These traps are ideal machines to locate on the drips of the steam chambers of multiple effects in place of the troublesome vacuum pumps, on account of their simplicity and automatic operation. They require very little space and a minimum foundation.

This trap is known as the Lytton perfect vacuum and lifting traps, and is made in sizes from 1 in. to 6 in. with capacities from ½ ton to 25 tons per hour, and manufactured by the Lytton Manufacturing Corporation, of Franklin, Va., with sales office at 50 Church street, New York.

The experiments which were so successful at the ice plant of the brewery were conducted under the direction of Mr. W. E. Parsons, Consulting Engineer, by the Chief Engineer of the Brewery and Mr. J. W. Lytton, President, and Mr. D. J. Lewis, Jr., Sales Manager, of the Lytton Manufacturing Corporation.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

REFINING METALS (*Continued*).

426,789, April 29, 1890, Moses G. Farmer, of Eliot, Maine.

Relates to an apparatus for depositing copper ingots from copper matte or other impure copper, used as anodes, upon a revolving drum having a polygonal depositing surface, and removable insulating separators along the angles between the several facets so that square ingots of copper will be formed thereon. The drum is intermittently rotated by a pawl and ratchet device. The drum cathode is made of a lead-antimony alloy, having a smooth cylindrical inner surface by which it is supported upon a roller in a suitable electrolyte. The removable insulating separators provide for the deposition of a continuous band of copper uniting all the ingots, to prevent them from falling off; the ingots may be readily removed by cutting this band.

465,525, December 22, 1891, Edward S. Hayden, of Waterbury, Conn.

Relates to an apparatus for the deposition of metals, and resembles the apparatus used in the process covered by his prior patent No. 377,487. The apparatus consists of a number of tanks, each having grooved vertical pieces of wood on the sides to permit the insertion of crude copper plates, leaving a space underneath the plate for the circulation of the electrolyte. The anode, of the metal to be treated, is at one end of the tank, and the cathode at the other, the intermediate plates being secondary electrodes. The electrolyte circulates through the tank, or tanks in series if electrically connected in series, and in multiple if electrically connected in multiple. The electrolyte may be sulphate of copper. A storage tank for the supply of solution is provided with a steam coil to regulate the temperature of the electrolyte.

467,350, January 19, 1892, Otto Stalmann, of Anaconda, Mont.

Relates to apparatus for refining metals, and consists of two concentric tanks having insulating material, such as asphaltum, between to prevent leakage. The electrodes are made of a plate of crude metal, serving as an anode, a layer of insulating material, and a superimposed sheet of pure metal serving as a cathode and electrically connected to the crude metal. These compound electrodes are used as secondary electrodes, between an anode of crude metal, and a cathode of pure metal, in the tank above noted. The crude metal from one compound electrode is dissolved and deposited upon the pure metal sheet of the adjacent compound electrode. Sulphate of copper is used as the electrolyte. The secondary electrodes are suitably supported in the tank, leaving a space below for the collection of slimes.

467,484, January 19, 1892, Otto Stalmann, of Anaconda, Mont., assignor to Marcus Daly, of same place.

Relates to process and apparatus for refining copper, and consists in detachably connecting an anode of crude copper to a cathode of pure copper, suspending a plurality of them as secondary electrodes in a vat containing sulphate of copper. A pair of connected elements are separated by insulating knobs, such as glass, from adjacent pairs. Instead of connecting the two plates, they may be separately suspended, and then short-circuited by wires. The electrolyte is maintained in circulation.

484,416, October 18, 1892, Charles R. Fletcher, of Boston, Mass.

Relates to an apparatus for electrolytically refining metals, and consists in clamping plates of sheet copper to serve as cathodes, to plates of crude metal, as anodes, and using the compound plate as a secondary electrode. The anode is cast with curved edges, its attached cathode is both longer and wider than the anode, and also has edges which curve in the same direction as those of the anode. When placed in the electrolytic vat, the compound plate is so inclined to the vertical that the upper face of the plate is the anode, and its under face the cathode. By using such cathodes, so disposed as to the anode of an adjacent plate, it is stated that the "growths," or "warts," which form on cathodes are avoided. Electrolyte is squirted over the surface of the anode from a series of jets having flattened nozzles, in order to wash off the impurities; this is said to reduce the liability of the anode-surface to becoming honeycombed.

485,618, November 8, 1892, Franklin Farrel, of Ansonia, Conn.

Relates to the refining of crude metals by electrolysis, using the secondary electrode process. The electrodes are not supported in grooves in the side of the tank, but are preferably suspended by tanks at the top of the electrode resting in hooks of insulating material. By this means all the crude metal is dissolved, leaving none to be remelted, thereby avoiding the expense of continually reworking a certain percentage of the metal.

489,632, January 10, 1893, Frank Gruessner, of Chicago, Ill., assignor of one-fourth to Francis B. Badt, of same place.

Relates to regenerating electrolytes used in refining copper. In time the copper sulphate solution becomes impure from the presence of arsenic, antimony, bismuth, and the like, originally in the crude metal. It is stated that suitable quantities of metastanic acid, if boiled with the electrolyte, will combine with and precipitate the arsenic as an insoluble salt; and also that it probably removes some other foreign substances.

Book Review

The Colloidal and Crystalloidal State of Matter. By Dr. **Paul Rohland,** Professor at the Technische Hoschschule, Stuttgart. Translated by W. J. Britland and H. E. Potts, M. Sc. 12 mo (12 x 19 cm.), 54 pages. Price $1.25 net. New York: D. Van Nostrand Company.

The book attempts to give broadly and from a most general standpoint, the essential modern ideas in this field of research and speculation. The attempt is crude; many statements are too strong, others too indefinite, others almost meaningless. Part of this may possibly be due to literal translation, a following too slavishly of the literal German order of expression of ideas; but most of what is complained of is certainly inherent in the original. The publishers co-operated toward the weak features of the book as far as they could—by taking on 39 pages of advertisements of their publications to a book containing only 54 pages of text.